A Semi-Monthly Technical Newspaper

Metallurgical & Chemical Engineering

Volume XIV, Number 4 February 15, 1916 McGraw Publishing Co., Inc.

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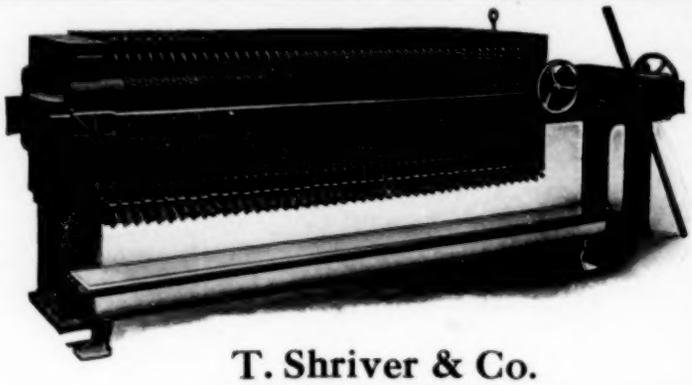

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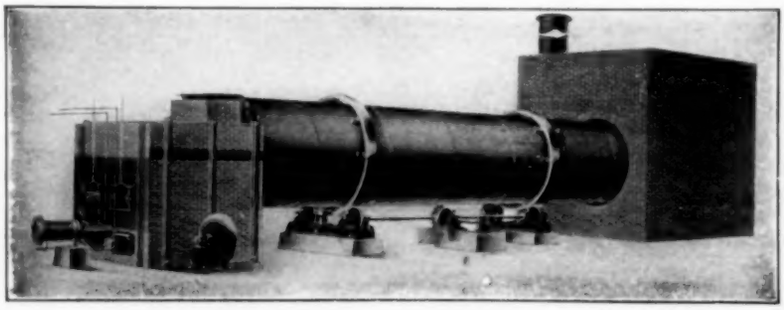

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New York Meeting of the American Institute of Mining Engineers

For the past few years the meetings of the American Institute of Mining Engineers have formed a string of grand successes—Great Falls and Butte, Salt Lake City, San Francisco—and in September there will be the Arizona meeting. But in the present week New York City has the privilege of welcoming the mining men from near and far. The outlook for a very profitable and enjoyable meeting is excellent. New York will spare no effort to make its visitors feel at home—and they will find that at present with the new subways in the making, the metropolis at the Hudson is in places nothing more than just a big mining town.

Our Noteworthy Pig Iron Production

The United States is now making more than one-half the world's pig-iron production. While pig-iron production outside the United States is less than before the war, the production of the United States is at a rate 25 per cent greater than the average rate in the record calendar year, 1913. Production of pig iron in the United States compares with the world's production as follows, the figures representing gross tons of 2240 pounds:

	United States	World	United States, Percentage
1850	563,755	4,400,000	12.8
1890	9,202,703	27,000,000	34.1
1900	13,789,242	40,200,000	34.3
1905	22,992,380	54,000,000	42.6
1910	27,303,567	65,300,000	41.7
1911	23,649,547	63,300,000	37.3
1912	29,726,937	72,600,000	41.0
1913	30,966,152	78,000,000	39.7
1914	23,332,244	60,000,000	38.9
1915	29,900,000	60,000,000	49.8

The above figures illustrate quite a story. From 1850 to 1890 there was what then appeared to be a tremendous growth in the iron industry, and the United States, then a very new country, naturally witnessed the greatest growth within its own borders. In the decade following there was no lack of either energy or brains in the United States, for there was noteworthy technical progress, but during the major portion of the decade there was an industrial depression in the United States, which prevented the country from doing more than maintaining its proportion of the world's production. Then there began a race between the United States and Germany, the honors going to the United States for several years, but Germany afterwards expanding more rapidly in production, whereby the percentage of the United States was sensibly reduced. The production of the United States in 1913 was very large, but not sufficient to result in as high a percentage as had been reached in 1910 or 1912. The year 1914 comprised seven months of peace and five months of war. The output of the

United States was low throughout the year, from industrial conditions, while the output of the European countries was greatly reduced in the early months of the war, the net result being that production in the United States, while greatly reduced, comprised almost 40 per cent of the world's production, which was so much the worse for the showing the rest of the world made. The proportion of the United States in the first seven months of 1914 was much less than 40 per cent. In 1915 the world outside the United States recovered part of the ground lost, producing iron at a rate considerably in excess of the rate in the early months of the war, but still far below the rate obtaining before the war. The production of the United States in 1915, although 3.5 per cent less than that in the record calendar year, nevertheless comprised practically one-half the world's production, against 43.4 per cent (not shown in the above table) in 1906, the highest proportion previously attained by the United States.

At the present time the United States is still farther in the lead, with production at the rate of 39,000,000 tons. The present rate of production outside the United States cannot be precisely estimated, but it is certainly very much less than 39,000,000 tons. It may not be at much more than 30,000,000 tons, the average rate in 1915, but taking it at 33,000,000 tons, which seems to be rather a high estimate, the proportion of the United States would be 54 per cent, a proportion which nothing but the war, probably, could have brought about. It seems improbable that either during the war or for years afterwards the United States can make a larger proportion of the world's pig iron than it is making at the present moment.

Smelting Bolivian Tin in the United States

The Smelting and electrolytic refining of Bolivian tin ore by the American Smelting & Refining Company at Perth Amboy, N. J., will remove "one of the most remarkable incongruities in the commercial relations of the United States and South America," to quote the Minister from Bolivia to the United States. Bolivia is one of the few countries in the Western Hemisphere that produces tin ore in large quantities. The United States, on the other hand, is the largest consumer of tin in the Western Hemisphere, importing annually from 45,000 to 50,000 tons but producing a negligible quantity. Nevertheless Bolivian tin ore has hitherto gone first to England or Germany for smelting and refining, and then brought to the United States, the latter country paying the cost of European production and transportation. This anomalous condition will now be changed; for the American Smelting & Refining Company has contracted for the delivery of 750 tons of high-grade Bolivian tin ore per month to be smelted at Perth Amboy, and the new plant will gradually be brought to its capacity of 1200 tons per month. The electrolytically refined product will be equal in quality to the best Straits tin.

The economic importance to this country of this change is only realized by a study of the value of Bolivian imports and exports and the small part which the United States has previously played in this business. According to the Department of Commerce, Bolivian imports in 1912 were valued at nearly $20,000,000, of which the United States supplied only $1,787,321 worth, as against $6,423,802 for Germany and $3,528,041 for the United Kingdom. In the same year the latter nation bought over $26,000,000 worth of Bolivian exports, while the United States was a buyer to the extent of only $152,582, or less than one-half of one per cent of the total. With the establishment of a tin smelting industry in this country Bolivia will have a local credit of millions of dollars per annum, and it must inevitably follow that our products will flow to Bolivia in greatly increased quantities.

The Bolivian tin industry was in unsatisfactory condition following the outbreak of war in Europe. Exports fell off and prices declined. The Bolivian government, therefore, welcomes the opportunity for closer commercial relations created by our purchase of one of her principal mineral products. Commercial treaty relations are satisfactory, and no unfavorable tariff is to be adopted by Bolivia that would affect the export of tin to this country. On the whole, the establishment of this tin industry promises to be an entering wedge for wider commercial relations with South America that should be of lasting benefit.

The Chemist and the Banker

In Dr. Baekeland's Perkin Medal address published in our last issue he put forward as an axiom: "Commit your blunders on a small scale and make your successes on a large scale." He then brought out the mental unpreparedness of American bankers to embark in chemical enterprises by the only wholesome and reasonable method of beginning small and growing large, to accord with this maxim.

The reason for this is that our bankers do not know enough chemists and our chemists do not know enough bankers. Of course, the banker is unwilling to risk good money on an experimental lay-out to test some chemical theory that "nobody" can understand. One might as well ask the chemist to invest his savings in cases of hosiery that are said to be a great bargain. He doesn't know the hosiery market and he would be afraid to take the stuff as a gift—and pay storage charges on it. And to any chemical proposition the banker is likely to say: "I don't know anything about chemistry," while his acquaintance with the family doctor is close enough to persuade him that he does know something about medicine and surgery.

Chemists and bankers do not know each other for a social reason. They have not been generally introduced, and when they have met they have not had things to say to one another that were interesting. Now there is no source of information too humble to be worthy the attention of him who seeks knowledge, and in this connection it might do no harm to observe the arts of the social climber who desires to break into society when she moves to town. She may be a very

vapid person, and exceedingly foolish, but she must use her wits to succeed. We have, in this example, the phenomenon of a human animal employing its wits to accomplish the fulfillment of its desires, which is always an interesting one. She lays her plans and makes her arrangements so that, to all appearances by accident, she is introduced to the most influential woman, to the dominant dowager of the temple of fashion. For this occasion she has prepared herself and trained like an athlete. By word or deed or by any of the innumerable wiles of women she engages the interest of the old lady so that the old lady will want to see her again. Following this up she plans her campaign to get more and more acknowledgments of acquaintance until, in time, authority creeps in. Then the dowager had better look out or the climber may climb above her and proceed to rule in her stead.

Far be it from us to suggest that chemists should lure bankers by tea parties and dances and auction or try to do so by spring hats or Chemists' Club coats, or even jewels galore. The instruments must be different, but there is a cousinship in method if they are to succeed. Bankers are on the lookout for investments just as the social dowager is looking for a larger kingdom. The dowager knows there are pitfalls, and that she must beware lest she give credence to persons who will not keep within the social traces. In like manner the banker is frequently lured by promoters anxious to get money into their enterprises, and the bankers' reputations depend upon their ability to steer clear of pitfalls. Now we venture the suggestion that chemists have not trained themselves to display their wares to the critical eye of the banker with the same care and wit that the social climber has availed herself of in breaking into society.

We know something of the wiles of the promoter, but the banker knows them a great deal better. He has juggled with figures and "conservative estimates" of profits anticipated, and prospective balance sheets until the banker has had his fill of that sort of business. He can cast up an imaginary balance sheet himself if he wants to. The attitude of mind that frightens him away most quickly is certainty of success by steps that he cannot understand. What he would like to know is the trade situation, manufacturing costs, the processes that have been already tried out and standardized, and those that are available but which have not been tried out and standardized. Give him this information so that he feels that he knows it and that he can clear up any doubts he may have in his mind to his own satisfaction by further investigation, and he will be as ready to go into chemical ventures as he is to underwrite a street railway enterprise or a mechanical industry. So far he has not been able to satisfy himself. This he must do for conscience' sake, and there are a thousand things that stand in the way of his innocent gaze. There is the tradition that chemical factories explode. Chemicals are poisonous. There is the danger of patent lawsuits. There is the problem of getting the best "formula"—one of the earliest questions that the banker is likely to ask. The formula or prescription theory is locked very tight in his mind. The problem of opening the banker's mind is often baffling to the wits. So far the American banker has not been able to satisfy himself. The German banker has, and that is the difference —the difference that we have to meet. It is not a question of ability, American bankers are as able as any; it is a question of habit. Bankers all over the world are like the social dowagers (who are often their wives), in that while they may be led, they will not be driven. They are greatly influenced by fashions and styles in investments. It is useless to warn them that they are losing great opportunities. The promoters have worked that plan of persuasion so often that it does not impress them any more.

So, there are the bankers, ready and willing to go into chemical enterprises if only somebody they know and trust can show them the real situation. They do not speak or understand the chemists' language, and they have no chemical affiliations because nobody has set the fashion for it yet. As soon as it becomes known that one or two leading banking houses have as good and competent connections in chemistry as they have in law or in civil engineering, others will be as ambitious to follow the lead as are the women to have their skirts cut short, now that the style calls for it.

The problem, then, is to get this fashion into vogue, and since the bankers must start it, and chemists are not bankers, the method of induction would seem available. For this purpose we suggest that the meetings of banks' and bankers' associations offer the opportunity. Like all of us, bankers are anxious to learn, but they have an idea that information must be imparted to them in a language that they can understand. This idea or prejudice must be respected, otherwise they will close their ears. The committees on arrangements of the meetings of the local, State and national banking associations are eager to secure addresses that will be of interest to those present, and there are hundreds of opportunities every year for men who are enlightened in chemistry to tell them of the possibilities of chemical development within their territory. A few scatterbrained enthusiasts who say more than they know could kill these opportunities for years to come, but a number of the right men, talking to bankers on the occasions when we know that they would be glad to hear them, would do a world of good. The work should be looked upon as pure missionary effort, with no thought of reward for the speaker. In this attitude of mind he has but two rules to heed: That he draw no pictures that cannot be realized unto success under good management, and that he go through his address with a fine tooth comb and clean out every technical expression that he can. The man of books and ledgers and bills of exchange and warehouse receipts must be made as familiar with every term used as he probably is with kilowatt-hours and surely is with bales of cotton or tons of hay. He will not object if the speaker explains something that he already knows; he will like it; it will prove that the speaker is sound in knowledge and understanding. It is no insult to any audience to speak in words of one syllable.

Reader's Views and Comments

Hydrometallurgy of Zinc and Lead

Electrolytic Zinc

To the Editor of Metallurgical & Chemical Engineering:

Sir: Replying to the discussion by Mr. C. A. Hansen of our article on the hydrometallurgy of zinc and lead in 1915, we are gratified that our contribution was the means of bringing out definite data on such an important subject.

As to Mr. Hansen's statement that we were slightly inaccurate in attributing to him the use of only the "hydrate" process, at the time of writing our paper we were in possession of data on his more recent work, but as it had been given to us confidentially, we did not feel at liberty to mention it.

As regards the hydrate process in general, the idea of neutralizing the acid generated during electrolysis by some device or other was probably introduced into the United States some years ago by Carl Hering. Hering's patents were taken up by a California concern and the production of electrolytic zinc has been experimented on for many years in California. Most of the work done in that State was based upon the belief that the acid generated during the electrolysis must be neutralized, and consequently the ideas developed in connection with that work have shown great ingenuity in the different methods by which this object could be accomplished. For this reason we stated that we regarded the electrolytic work which has been done in Ontario as being very similar to that which has been done in California, and especially at Bully Hill. However, this is no reflection on the work which has been done there, as we believe that the work of Mr. Hansen and his associates has been a most important contribution to the commercial production of electrolytic zinc.

We note Mr. Hansen objects to our statement that an electrolytic zinc plant is not a poor man's proposition, and offers as evidence that we are in error regarding his estimated cost of a plant for reducing 200 tons of spelter per day from the Butte and Superior concentrates as being $2,400,000. His estimate is doubtless very close to what the actual cost would be, and it was just this fact upon which we based our own conclusions. Perhaps it would have been better had we stated that the cost of an electrolytic plant is about the same as that of a retort plant for the same capacity. However, there are few companies in this country which can afford the outlay for an electrolytic plant. Hence our statement that such a plant is not a poor man's proposition, and our belief that at the present time the electrolytic zinc process is not well adapted to the production of zinc by small companies who wish to work the ores from small low-grade deposits, or even small high-grade deposits. An electrolytic plant calls for large deposits of favorable ore, and a steady production.

We believe that Mr. Hansen's communication constitutes a very important contribution to the literature of the subject of electrolytic zinc.

Dorsey A. Lyon.

Salt Lake City, Utah.

Electrolytic Lead

To the Editor of Metallurgical & Chemical Engineering:

Sir:—Replying to the communication from Mr. Henry R. Ellis regarding what we had to say as to the hydrometallurgy of lead, we note Mr. Ellis claims to having had a prior knowledge of the facts underlying the brine process. We had not intended to give the impression in our paper that we were presenting new facts unknown to previous writers, but rather that we had worked out a new process, based on facts worked out in three semi-commercial plants and in our own laboratories.

As a matter of fact, Mr. Ellis in his turn was not the original discoverer of the fact that lead can be extracted from its ores by the use of a strong brine. Several months ago in looking up the literature on this subject we found that in 1854 M. M. Bequerel of Paris had carried on experiments based upon the above facts. His work was reported in *Comptes rendus* (1854), 38:1095, and also in *Dingler's Polyt. Jour.* (1854), 133:213. Bequerel treated over 20,000 lb. of ore from Mexican and South American mines, as well as small lots of ore from the Erzgebirge, the Altai Mountains, and from different localities in France. He gave these ores a sulphatizing roast in the presence of salt, then leached out lead sulphate and silver chloride with a saturated brine. The brine was electrolyzed to recover the metals. This proposal of Bequerel's is almost identical with our present proposal, with the exception that modern commercial conditions are different from those existing at the time of Bequerel. As a source of electric current Bequerel only had primary cells, and it was doubtless the high cost of this item which caused his work to be dropped.

The process described by Mr. Ellis is different from the process mentioned by us in that he does not use a saturated solution of salt. We have found that the solubility of lead compounds falls off very materially as the solution is diluted, so that a half saturated solution has only 1/7 of the dissolving power of the saturated solution. Mr. Ellis, however, used hot solutions, which are capable of dissolving more lead than cold solutions, but we prefer the use of a cold saturated brine, which we feel is much closer to being commercially practicable.

Dorsey A. Lyon.

Salt Lake City, Utah.

Hydrometallurgy of Lead

To the Editor of Metallurgical & Chemical Engineering:

Sir: Your correspondent Henry R. Ellis is the author of a statement in your issue for Feb. 1 that lead sulphate was extracted with brine in 1892. It is known that the process was employed earlier than that, as an English patent (No. 2316) was granted to a Mr. Wright in 1869 for identically the same operation, and it is not a long guess that a thorough search would reveal references to it in Pliny and Agricola.

There were two features in the work referred to by Lyon, Ralston & Cullen in your issue for Jan. 1, which apparently have escaped the observation of experimenters during the last fifty years. A diligent search has so far failed to discover any reference to them and there is accordingly still room for hope that they may have the element of novelty—but that is another story.

S. A. Ionides.

Denver, Colo.

More Newspaper Science

To the Editor of Metallurgical & Chemical Engineering:

Sir:—The accompanying literary and scientific gem, clipped from a Western newspaper, sheds new light on copper hydrometallurgy as practiced by the energetic

and enterprising reporter. Speaking in detail of a certain process, he says:

"The insoluble compounds in the solution, such as oxides, come over in the stream of electrolysis and are deposited on the cathode plate in the form of pure copper and traces of other metals that may be present."

Thus another triumph is scored for electrometallurgy, and the gaiety of metallurgists proportionately increased. We are left in the dark, however, as to just how this "stream of electrolysis" is measured or controlled. How unfortunate that "pure copper" thus produced should be contaminated with "traces of other metals that may be present!" Something should be done to overcome this slight defect, and the point is brought to the attention of your readers in the hope that so valuable a process may shortly be perfected.

Denver, Col. H. C. P.

Western Metallurgical and Chemical Field

Pacific Coast Enters Electrochemical Field

The State of California is shortly to have a new industry.

The Great Western Electrochemical Company has recently been organized and incorporated in the State of California, for the purpose of manufacturing electrochemical products.

The first operation will be an electrolytic caustic soda plant in which caustic soda and chlorine products will be manufactured. The Moore Allan cell will be used, and bleach chambers such as used at the Dow Chemical Co. will be installed.

The future plans for this company are still somewhat indefinite, but it can be stated with certainty that other electrolytic products as well as electric-furnace products will be manufactured.

Ground is already broken for the new factory and the buildings and equipping of same is in charge of the Foundation Company of New York.

The location of the factory in Pittsburgh, Cal., 50 miles from San Francisco, is a very favorable one. The site, comprising some 40 acres, faces on the San Joaquin river which is navigable at this point by sea going vessels. In addition, service by the Southern Pacific and Santa Fe railroads is obtained.

The power supply is furnished by the Great Western Power Company, whose main transmission line passes in the close vicinity of the factory site.

The necessary salt and lime will be obtained from local supplies, and the needed fresh water is obtained in abundance in the San Joaquin river.

The plant will be in operation by July, 1916.

Directors of the new company are:

Mr. John F. Bush, formerly General Manager and Vice-President of Hooker Electrochemical Company.

Mr. Franklin Remington, President of The Foundation Company, New York.

Mr. J. W. Beckman, formerly of Niagara Falls, N. Y.

Mr. L. Bowen of Detroit, Mich.

Mr. Mortimer Fleishhacker of San Francisco, President of the new Electrochemical Company.

Mr. Louis Bloch, San Francisco, General Manager and Vice-President of Crown Willamette Paper Company.

Mr. Herbert Fleishhacker, President of Anglo-London Paris National Bank, of San Francisco.

Mr. A. Lilienthal, Secretary of the new corporation.

Mr. C. Hall.

The firm of Bush, Beach & Gent, 80 Maiden Lane, New York, will be the selling agents and will also maintain an office in San Francisco.

Annual Meeting Canadian Mining Institute

The eighteenth annual meeting of the Canadian Mining Institute will be at Ottawa, March 1, 2 and 3, 1916. The headquarters will be at Chateau Laurier Hotel. A symposium of flotation will be given by T. A. Rickard, E. P. Mathewson, Henry E. Wood and H. W. DuBois. Another feature of interest will be a paper on rock-crushing tests at McGill University, by J. W. Bell. The subject of education in mining and metallurgy will also receive attention from several contributors. The annual dinner will held on the evening of March 2, and on other days of the meeting visits will be made to places of technical interest in Ottawa.

Great Falls Electrolytic Zinc Plant

Work on the construction of the electrolytic zinc plant of the Anaconda Copper Mining Co., at Great Falls, Mont., is now in progress and will be completed as rapidly as possible, according to *The Anode*.

This plant will take the concentrates from the concentrator at Anaconda and will produce 100 tons of electrolytic zinc per day. The operations to be performed may be said, briefly, to be as follows: The concentrates are given an oxidizing roast, which renders the zinc and a small amount of other minerals soluble in a sulphuric acid solution. The roasted concentrates are, therefore, leached by such a solution and the impurities are precipitated by adding limestone and zinc dust as necessary. The purified solution is then sent to the electrolytic cells and the zinc is deposited as a sheet on the cathode.

The electrolytic zinc made by this process is much purer than spelter and has certain physical properties which spelter does not possess. Among other things, it is much more ductile and is particularly suited for use in the manufacture of high-grade brass.

Part of the slimes flotation plant building is being equipped with apparatus to treat 200 tons of zinc ore per day by the oil flotation process. The concentrates made in this plant will be treated at the Anaconda experimental zinc plant, which is to be enlarged, and will produce 25 tons of zinc per day.

Plans are now being drawn for a concentrator to treat 2000 tons of zinc ore by the oil flotation process, and it is expected that the plant will be completed by July.

The concentrates from this plant will be sent to Great Falls for treatment at the electrolytic zinc plant at the B. & M. Reduction Works.

Terms of Settlement of Arizona Labor Strike

After four months of idleness, during which time the employees of the Arizona, Detroit and Shannon copper companies lost their wages, and the companies lost the profit from about 6,000,000 lb. of copper per month, the labor strike in the Clifton-Morenci district of Arizona has been settled. The net result is a loss for all concerned, and the terms of settlement finally accepted are such as could have been obtained in the beginning.

The conditions under which work has been resumed are those offered by the managers of the several companies on Jan. 8, and are substantially as follows: The charters of Western Federation of Miners locals in Clifton, Morenci and Metcalf are surrendered in good faith, and the influence of outside agitators is eliminated. The men return to work under the former wage scale, which the managers offer to extend to meet the increased price of copper. It is understood that all workmen employed by the companies will receive full and adequate protection. The managers agree that no individuals of any nationality will be barred from reemployment because of their connection with the strike movement, except those who have been guilty of acts

of violence; and for this latter reason no more than ten will be discriminated against by the three companies.

The managers agree that upon resumption of operations and conditions becoming normal, they will meet committees of their employees with a view to adjusting any grievances or considering any questions which the employees may wish to bring up, the intention of the managers being to make conditions as satisfactory as possible for the employees.

The table below shows the rates of wages received by the principal classes of employees for the period Aug. 15 to Sept. 15, 1915, together with the rates which will be paid to the same classes of labor when the indicated prices for copper prevail.

		Wages, with Copper Selling At					
		18c.		20c.		24c.	
Occupation	Hours	Per Hour	Per Shift	Per Hour	Per Shift	Per Hour	Per Shift
Miners	7½	$0.36½	$2.74	$0.39½	$2.96	$0.45½	$3.41
Muckers and trammers	7½	.28½	2.14	.31	2.33	.36	2.70
Timbermen	7½	.38½	2.89	.41	3.08	.47	3.52
Laborers (surface)	8	.24	1.92	.26½	2.12	.31½	3.52
Charge-wheelers	8	.26½	2.12	.28½	2.28	.33½	2.68
Skimmers	8	.44	3.52	.46½	3.72	.51½	4.12
Punchers	8	.33	2.64	.35	2.80	.40	3.20
Laborers	8	.24	1.92	.26½	2.12	.31½	2.52
Carpenters and electricians	8	.54	4.32	.57½	4.60	.62½	5.00
Machinists and boilermakers	8	.57	4.56	.60½	4.84	.66½	5.32

Non-Ferrous Metal Market

Feb. 10.—The metals continue in their strong position. The only decline recorded during the last two weeks was one of about ½c. in tin. The other metals either remained unchanged or registered advances. The extraordinary situation in copper has forced that metal still higher, while most of the other metals are scarce to a greater or less degree for early deliveries.

Copper.—Producers are sold up for several months ahead, in some cases up to July and August and deliveries this side of May or June are hard to obtain. The market has risen steadily since our last report and quotations are now 27 to 28c. for nearby deliveries and 26 to 27c. for June and later deliveries. Conditions abroad are reported to be about the same as ours and a scarcity has forced prices up. Electrolytic is quoted in London at £131 spot. Rumors are afloat of heavy foreign buying for shipment now and after the war. England is said to have bought 100,000 tons and Germany a large amount for later shipment. Whether the present market is due to these purchases is a matter of conjecture, but it is thought by a number of authorities that this is the case. At any rate the domestic consumption is very heavy and articles in which copper is used will naturally be higher priced.

Tin.—Good sales of tin were reported the last few days in January. Early deliveries and futures have not been hard to buy. Stocks on Jan. 31, figured from the deliveries, were 2401 tons. Arrivals in January were 4430 tons. There is no scarcity of tin, either spot or en route. The market during the first ten days in February has been dull. Straits is quoted at 41.25 f.o.b.

Lead.—Up until Feb. 8 lead remained dull. On that day a good demand appeared for both prompt and future deliveries, and lots for sale by outside parties not connected with the Trust are commanding a premium. The Trust price is still 6.10 New York.

Spelter.—Buying of spelter was scarce up to Feb. 8, when large purchases were made. Deliveries up to May are being sold. Second quarter deliveries are quoted at 16.50c. St. Louis, March at 18.50, February at 18.75. Spot spelter is 19c. at St. Louis and 19.17½ to 19.42½ at New York.

Other Metals.—Aluminium advanced 1c. on good inquiry and is now quoted 54.00 to 56.00. Antimony was in good demand on Jan. 31, after two weeks' dullness. Supplies are scarce and the quotation is now 43.50 to 44.00, with the market firm. Silver moved up and down within a narrow range and is now quoted at 56⅝. Platinum is still hard to quote, owing to the scarcity, and $88 to $100 per oz. remains the nominal price. Quicksilver is quoted at $275 to $300 per flask.

The Iron and Steel Market

The steel market has become distinctly quiet as to strictly new buying, but on the whole there is as heavy specifying as ever on open contracts. In some quarters the slowing down in the rate of new buying is interpreted as suggesting that a change has occurred in the general situation, but the majority view is that the slowing down is due to the physical condition that the market is already sold out.

The future of the mills for the purpose of forecast may be divided into two periods, that for which they have specific orders on books, or orders practically assured, and the later period for which consumptive demand must develop, on the basis of higher prices than have hitherto obtained. With the average mill the first period comprises about six months. Some mills have definite orders and specifications on books for fully six months, while others have such business for a shorter period, with every indication that orders will continue heavy for a time. The great bulk of this specific business is at prices much below those now ruling. There are some specifications yet to be filled that were filed against fourth-quarter contracts, and the contracts now being specified against are as a rule at much lower than current prices, averaging in the case of bars, plates and shapes about 1.45c., while the present market is 2.00c. on bars and shapes and 2.10c. on plates.

It is natural that doubt should be entertained whether consumption will be seriously curtailed in future when buyers must pay the present high prices. It is a question not for nearby months but for the second half of the year, perhaps only for the fourth quarter. The prospect for heavy consumption is made favorable by the fact that the present demand is very well distributed, and appears to be chiefly for the filling of everyday requirements rather than for large construction projects in which costs are carefully considered. It is much more difficult than usual to trace to its ultimate use the steel being shipped. The consumption of standard rails is extremely light and structural tonnage is far from heavy. There is feverish activity among builders of machinery and automobiles, lines of consumption in which price cuts almost no figure. It is interesting to observe that there is but one steel commodity that has not advanced in price, standard steel rails for domestic roads, and it is that commodity which is being produced at the lowest rate. Rails for export have been selling at relatively high prices, and rail exports are at a record rate. In this one instance at least the price has no influence upon the demand.

Exports of iron and steel returned by weight were 363,000 gross tons in November, slightly more than in October, but less than in any of the three months preceding. There is no reason to suppose that the consumption of steel in the manufacture of export products that are not returned by weight has increased from the average of the months of July to November inclusive. In that period the exports that are returned by weight represented a rate of about 4,000,000 tons of finished rolled steel per annum. An estimate

of 3,000,000 tons of finished steel a year would probably be high for that consumed in the manufacture of the indirect exports, including loaded and unloaded shells, machinery, automobiles, railroad rolling stock, etc., so that it does not appear that finished steel made for all export purposes is at a rate higher than about 7,000,000 tons at the outside. The current production is at approximately 29,000,000 tons.

The steel trade is so extremely prosperous at present that interest centers not so much upon the bearing that current events may have on the present or the nearby future as upon the bearing they have upon the course of trade in the more distant future. Obviously the steel mills will be under pressure for more than six months, but whether they will be under pressure for nine months, a year or three years is another matter. Many of the mills are uncomfortably oversold at present and would require months even of a quiet market to recover. What is probably an important indication for the future is that at the present moment the demand for prompt material, which commands premiums over forward deliveries, is as great as it was 30 days ago and greater than it was 60 days ago. While the large mills do not accept premiums for prompt shipment, the smaller mills make this their rule. In plates, for example, carloads for shipment within a week or two are bought freely at 3.00c., while orders for 100 to 500 tons are common for delivery over 90 days, at about 2.50c., these prices comparing with the regular price of 2.10c. made by large mills for delivery at their convenience.

Pig Iron

The dullness that characterized the pig-iron market in January has since developed into absolute stagnation, with signs in some quarters that prices are perhaps a shade easier, though they are not quotably lower. Pig-iron production by both steel interests and merchant furnaces has increased further, and is now at the rate of about 39,000,000 tons per annum, a rate which a year ago no one thought the furnaces could possibly compass. The question is the one which so often develops in the pig-iron market, which party, the buyers or the sellers, can hold out the longest. We repeat former quotations: No. 2 foundry iron, delivered Philadelphia, $19.75 to $20.25; f.o.b. furnaces, Buffalo, $18 to $18.50; delivered Cleveland, $18.80; f.o.b. furnaces, Chicago, $18.50; f.o.b. Birmingham, $15; at valley furnaces, 95 cents higher delivered Pittsburgh: Bessemer, $21; basic, $18 to $18.50; No. 2 foundry, $18.50 to $19; forge and malleable, $18 to $18.50.

Steel

There has been a slight softening in the unfinished steel market, to the extent that odd lots of billets and sheet bars can now occasionally be picked up for fairly prompt shipment, whereas until very recently there was practically no steel to be had except at prohibitive prices. While the market is still not clearly defined, it is probably represented better by $33 for Bessemer and $35 for open-hearth billets and sheet bars, f.o.b. maker's mill, Pittsburgh, or Youngstown, than by any other figures. We note sales of three lots of sheet bars aggregating about 1500 tons for prompt shipment on a basis of a trifle under $35, f.o.b. maker's mill, Pittsburgh, but it is only occasionally that such lots can be picked up. Rods are quotable roundly at $45, Pittsburgh, but are largely nominal at that figure.

Finished Steel

On Feb. 4 the Carnegie Steel Company advanced its prices on bars, plates, shapes and hoops by $2 a ton, the advanced prices readily becoming the general market. For early shipment plates are bringing 2.50c. to 3.00c., bars commanding 2.25c. to 2.50c. Regular mill prices are given below, f.o.b. Pittsburgh, unless otherwise noted, and are for delivery at mill convenience.

Rails, standard sections, 1.25c. for Bessemer, 1.34c. for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 2.10c.

Shapes, 2.00c.

Steel bars and bands, 2.00c., base; steel hoops, 2.20c., base.

Iron bars, Philadelphia, 2.409c.; Chicago, 1.90c.; Pittsburgh, 2.20c. to 2.25c.

Plain wire, 2.05c., base; galvanized wire, $2.75c.; wire nails, 2.20c. per keg, base; painted barb wire, 2.35c.; galvanized barb wire, 3.05c.

Sheets, blue annealed, 10-gage, 2.65c. for Bessemer, 2.75c. for open-hearth; black, 28 gage, 2.60c. for Bessemer, 2.75c. for open-hearth; galvanized, 28-gage, 4.75c.; painted corrugated, 28-gage, 2.80c.; galvanized corrugated, 28-gage, 4.80c.

Tin plate, $3.75 to $4 for 100-lb. cokes.

Steel pipe, ¾ to 3-in., black, 76 per cent off list; galvanized, 60½ per cent off list.

Boiler tubes (less than carloads), 3½ to 4½-in. 64 per cent off list; iron, 60 per cent off list.

Structural rivets, 2.70c.; boiler rivets, 2.80c.; cold-rolled shafting, 45 per cent off list; cold-rolled strip steel, 4.00c., base.

The Flotation Conference at the University of Kansas

At the suggestion of Mr. Dorsey A. Lyon, metallurgist for the United States Bureau of Mines, the first gathering of metallurgists and mining men to consider exclusively the rapidly growing subject of flotation came together on Jan. 28 and 29 in the University of Kansas at Lawrence. The Kansas institution was agreed upon as the first meeting place of the conference because of its central location and because of the pioneer work of its Division of State Chemical Research in investigating the application of flotation to zinc recovery within the State.

The underlying object of the conference was to discuss the problems which are yet to be solved in the process of flotation and more particularly to consider the research work now being conducted in various universities and mining schools with the object of encouraging further research and preventing useless duplication among the various investigators concerned.

Among the visitors in attendance at the conference were Messrs. D. A. Lyon and O. C. Ralston of Salt Lake City, representing the United States Bureau of Mines and the University of Utah; Dr. A. L. McRae, director, and Mr. Charles Clayton of Rolla, Mo., representing the Missouri School of Mines; Mr. R. C. Palmer of Madison, Wis., representing the United States Forest Products Laboratory, and Mr. C. A. Wright of the Bureau of Mines, stationed at Joplin, Mo., representing the Joplin Zinc District.

The University of Kansas was represented by Prof. W. A. Whitaker, director of the Division of State Chemical Research, which is prosecuting several flotation investigations; Mr. George Belchic, flotation investigator, and by Messrs. Roy Neal, S. F. Farley and L. E. Van Velzer, assistants in flotation work. Others present from the University of Kansas were Profs. E. H. S. Bailey, C. F. Nelson, A. C. Terrill, H. C. Allen, G. W. Stratton, W. H. Twenhofel, A. H. Sluss and R. L. Crider. Letters were read from more than thirty pro-

fessors of mining and metallurgy throughout the country expressing their approval of such a conference and regretting their inability to attend.

In addition to several informal conferences which were held on the 28th, two regular meetings were held on Saturday, Jan. 29, with Professor Whitaker acting as chairman. The morning session continued from 9.30 a. m. to 12.30 p. m. and the afternoon session from 2 p. m. to 5 p. m. The morning session was consumed by the three following papers and the resulting discussion:

1. "Some Things that May Be Accomplished by a Conference on Flotation," Mr. D. A. Lyon, United States Bureau of Mines.

2. "Proposed Research on Flotation," Dean Milnor Roberts, University of Washington (read by Dr. A. L. McRae). Discussion led by Messrs. Lyon and McRae.

In the course of his remarks, Dr. McRae stated that the Missouri School of Mines was engaged in making a complete bibliography on flotation and that this would be completed soon. Mr. Lyon urged the publication of such a bibliography.

3. "The Theories of Flotation," Mr. George Belchic, University of Kansas. Discussion led by Messrs. Ralston and Whitaker.

The discussion on this paper was very full and brought out several new points on theory, especially as regards the relation to flotation of colloid knowledge and surface tension phenomena. The theoretical and practical flotation questions now being investigated at the University of Kansas were reviewed. The discussion was succeeded by three informal talks: Dr. C. F. Nelson of the University of Kansas spoke on colloids; Mr. R. C. Palmer of the United States Forest Products Laboratory spoke of the desire of his laboratory to help in evolving the proper flotation oils as products of wood distillation; Mr. Charles Clayton of the Missouri School of Mines reviewed briefly some of the work which has been done in that institution.

The afternoon session was devoted almost entirely to the formulation of a questionnaire on theory, which, it is intended, shall be sent to all institutions that are engaged in investigating flotation problems. A set of questions was proposed by Mr. O. C. Ralston, and in the general discussion which followed the list was increased by the addition of other questions. At this time, Mr. Roy Neal of the University of Kansas gave some of the data which he had obtained in his experiments on surface tension and interfacial tension. The closing feature of the conference was a talk by Prof. A. C. Terrill of the University of Kansas on "Some Observations on Flotation Practice." Following this talk the conference went into executive session for a few minutes and then adjourned.

On Friday evening, Jan. 28, a "smoker" was given in honor of the visitors at the University Club, at which talks were made by Messrs. Lyon, Palmer, McRae and Whitaker.

American Electrochemical Society New York Section

Electrochemical War Materials — Electrolysis of Underground Structures

For the meeting of Feb. 11, Dr. C. G. Fink, chairman of the New York Section of the American Electrochemical Society, had arranged an unusually interesting, instructive and timely program, in the form of a symposium of papers on electrochemical war supplies. The attendance at the meeting was very good. Limitations of time and space force us to reserve the full report of this meeting for our next issue. The paper by Dr. G. Ornstein on electrolytic chlorine presented in this symposium is published in full on page 215 of this issue.

For the evening of March 10, 1916, the American Institute of Electrical Engineers has extended a courteous invitation to the New York Section of the American Electrochemical Society for a joint meeting in the United Engineering Societies Building. The subject of discussion will be electrolytic corrision of underground structures.

Necessity for an American Dye-Stuff Industry

At the third National Foreign Trade Convention held at New Orleans, La., from Jan. 27 to 29, Mr. HENRY HOWARD, vice-president of the Merrimac Chemical Company, and chairman of the executive committee of the Manufacturing Chemists' Association of the United States presented an interesting paper on "The Necessity of an American Dyestuff Industry to Aid Export Trade in Textiles."

The speaker pointed out that the stoppage of importation of dyes from Germany "has been disastrous to what we all desire—namely: an increase in our foreign trade in textiles. Many textile mills have been obliged to curtail their output at a time when, if we had been independent of other countries, with a dye industry well established, these same mills could have been running night and day manufacturing goods for export and establishing foreign connections by aid of which a considerable percentage of this business might have been retained after the close of the war."

The speaker stated that the fact that there is no adequate coal tar industry in this country can not be charged against the Democratic party but is due "to the strenuous effort of the foreign manufacturers and American importers, ably assisted by the short-sighted selfishness of these same textile manufacturers who are now the principal sufferers."

The speaker pointed out that the American dyestuff industry gave promise in 1882, but then came the fatal tariff reduction in 1883. Now the American textile manufacturers stand ready to co-operate in any way possible in order to establish a permanent coal-tar industry within the United States.

Mr. Howard compared the present condition of textiles in this country with that of iron and steel. "The latter industry, thanks to the consistent protection which for years was extended to every branch of the business, is now absolutely independent of the rest of the world in its ability to manufacture all standard grades of iron and steel products in this country, with the result that not only are all American users of iron and steel products getting their supplies as regularly as before the war began, but a large export business is being developed as the result of the paralysis of this industry abroad. In textiles the saving of a possible 4½ cents on a hundred dollars' worth of product has resulted in so throttling the textile industry that it is having hard work to supply our local markets, much less to gain a strong foothold in the foreign markets formerly supplied by the belligerent countries."

Mr. Howard considers that a permanent non-partisan tariff commission is necessary. Further, unfair competition of foreign dye manufacturers must not be permitted. Finally Mr. Howard pointed out the intimate connection between the manufacture of dyes and explosives. A complete coal-tar dye industry in this country is necessary for the sake of preparedness.

Mr. Howard was not alone in referring to the desirability of an increased export trade. Other speakers emphasized the stabilizing influence which export trade has in any industry.

The Calcination of Zinc Carbonate

BY WILLIAM P. SIMPSON

Calcination in general has been practiced as long as quicklime has been known; it consists simply in the conversion of carbonates into oxides by the application of heat.

In the particular adaptation to zinc carbonate the process involves heating the ore to such a temperature as will drive off carbonic acid without volatilizing zinc, leaving zinc oxide, with its decreased weight per unit of zinc, in place of the original carbonate.

Aside from better market value for ores of higher zinc content, the object of calcination may be one of several, depending upon the locality in which the operations are carried on or upon the subsequent treatment of the ore. In the former case the reduction of transportation cost effected by this form of concentration is the desired end. In the latter it is the lowering of fuel expense, decrease in bulk of material handled or the time element involved in the further working of the ore.

For instance, the first mentioned incentive may be illustrated by the case of calcination in Mexico, to accomplish a saving in freight charge on the long haul to the Oklahoma smelters. Illustration of the latter object may be had in Belgium and Germany, where calcination is employed before retort smelting, thus saving time in distillation, as otherwise a large part of the smelting period would be consumed in driving off CO_2 before distillation of zinc could begin. Further, the smelting of calcined ore would increase the capacity per retort due to the previous concentration of the ore by calcination. Formerly it was considered essential to get rid of CO_2 before retorting carbonate ores. Of late years, however, American smelters have charged crude carbonates into their retorts, proceeding with distillation as when roasted blende is used.

Labor Cost Limits Use of Process

Factors in the practicability of employing calcination in any given case are the relative cost of labor and fuel. As in the early history of sulphide roasting, so in the calcination of zinc carbonate, hand work has thus far given better results than mechanical treatment. This fact has retarded the use of calcination in this country, where labor is the big item in operating cost, fuel being comparatively cheap. In Europe labor is cheap and fuel expensive; hence at the expense of labor and the saving of fuel, calcination precedes retorting as previously explained, a cheaper grade of coal being used in calcination than for smelting.

The history of calcining zinc carbonate begins with the exploitation of such ore in Algeria, Greece, Italy, Spain and the island of Sardinia. In these countries calcination is practised at the mines. For many years the smelters of Belgium and Germany have employed calcination before retorting. In the United States the only locality where the process is being conducted, as far as I am aware, is at Arden, Nevada, where purely mechanical handling has thus far resulted in very incomplete, but profitable, calcination. For several years Oklahoma smelters have been receiving increasingly large shipments of calcined zinc ore from Mexico, and until the European war broke out Mexico had for ten years shipped calcined ore as ballast to Europe.

Beginning with the chemical or laboratory side of the subject, there is one ore which is of chief importance, viz., smithsonite, $ZnCO_3$. The molecular weight of this carbonate is 125.4; that of zinc oxide is 81.4. This gives a ratio of carbonate to oxide of 1:0.649, or roughly 1:0.65. Assuming the zinc in an ore assaying 32.5 per cent to be pure smithsonite, the zinc percentage after calcination will be 32.5 ÷ 0.65, or 50 per cent. Experience, however, shows that this method of determining the zinc content which may be expected from calcining any particular ore is not satisfactory. Several conditions explain why this procedure may not be confirmed in actual practice. First, the ore may not contain its zinc as pure carbonate; second, calcination is rarely complete, and a product is seldom obtained which would be comparable to what is known as a sweet roast in desulphurization, from 0.4 per cent to 1.5 per cent of carbonate often remaining in the best calcine. On the other hand, there are often carbonates other than zinc in the ore, such as calcium carbonate. The decomposition of these may cause a decrease in weight and consequent increase in zinc in the calcine beyond the percentage that would be indicated by this method of calculation.

Laboratory Calcination Assay

The relation of the calculated result to that obtained in actual practice may be determined by a laboratory calcination assay, as follows: Ten grams of the powdered crude ore is weighed into a 3-in. scorifier or saucer-like porcelain dish and spread in as thin a layer as possible. The dish is then placed in the muffle of an assay furnace at such temperature as will bring the ore to a dull red glow in ordinary daylight. The muffle door is left open and the furnace doors are so manipulated as to cause the greatest draft through the muffle and produce the best oxidizing conditions. A little practice and experience based upon comparative tests and assays will enable the assayer to gage the temperature by the eye. Too high a temperature, above 920 deg. C., results in loss of zinc; too low a temperature, below 300 deg. C., or too short a time, will result in incomplete calcination. Two hours in the furnace under proper conditions will give satisfactory results.

At the expiration of this time the assay is removed from the furnace, and when cool it is returned to the scale pan of the balance, with the same ten-gram weight on the opposite pan. Equilibrium is now established

TABLE I

Laboratory calcination assays, showing comparison of different methods of calculating percentage of zinc in the calcine.

No.	Crude Ore, Per Cent Zinc	Calcination Loss, Per Cent	Complement of Calcination Loss	Per Cent Zinc in Calcine Calculated from Calcination Loss	Per Cent Zinc in Calcine by Assay	Per Cent Zinc in Calcine if Ore Were Pure Carbonate (% Zn ÷ 0.65)
1	16.7	13.7	86.3	19.4	19.1	25.8
2	19.7	14.3	85.7	23.0	22.7	30.3
3	20.8	16.6	83.4	25.0	25.0	31.4
4	21.4	19.0	81.0	26.5	26.2	33.0
5	22.3	18.3	81.7	27.3	26.7	34.3
6	28.5	21.0	79.0	36.1	36.0	43.8
7	31.9	25.0	75.0	42.5	42.6	49.1
8	33.3	18.9	81.1	41.1	40.5	51.3
9	41.2	24.0	76.0	54.2	53.7	63.5
10	42.4	24.5	75.5	56.2	55.0	65.2

TABLE II

Ideal Charge

Ore Class	Quantity	Class Factor	Resultant
1	20 barrows @	4	80
2	5 barrows @	3	15
3	2 barrows @	2	4
4	1 barrow @	1	1
	28		100

TABLE III

Showing variation limits in charges and their effect on capacity. One barrow-load weighing about 400 lb.

Resultant	Charges of 28 Barrows Each	Capacity Tons
95	3½	20
97	4	..
99	4½	..
101	5	..
103	5½	..
105	6	..
107	6½	36

by adding the necessary small weights to the pan containing the assay, and these weights indicate directly the percentage loss of weight by calcination. This is called the "calcination loss." Now the complement of this calcination loss (80 per cent in the case of 20 per cent loss) divided into the zinc percentage of the crude ore will give a very close approximation to the zinc percentage that may be expected from calcination on a commercial scale. The actual operation in kilns generally results in still greater calcination loss than the assay shows.

Table I contains the results of ten laboratory calcination assays, and shows comparatively the zinc content of the calcine as determined (*a*) by the method of calcination just given; (*b*) by actual assay of the calcine; and (*c*) by calculation on the assumption that the ore is pure carbonate. The first two sets of figures agree quite well and show the reliability of the calcination assay. The third set, on the other hand, show by contrast the misleading results obtained when assuming the zinc to exist as pure carbonate.

In practice, daily samples of crude ore and calcine are assayed for zinc and calcination loss, to control the operation of the furnace or kiln, just as in lead smelting assays of bed samples and slags inform the metallurgist of the operation of his blast furnace.

Three Essentials to Success

Turning now to the practical application of the process, the method and equipment are very simple; yet three points are of such vital importance that calcination has proved a failure in the hands of novices. These three points are: 1—sizing of the ore; 2—form of furnace or kiln; 3—manner of charging.

For calcining in kilns the ore is passed over grizzlies or screens to produce the following four grades:

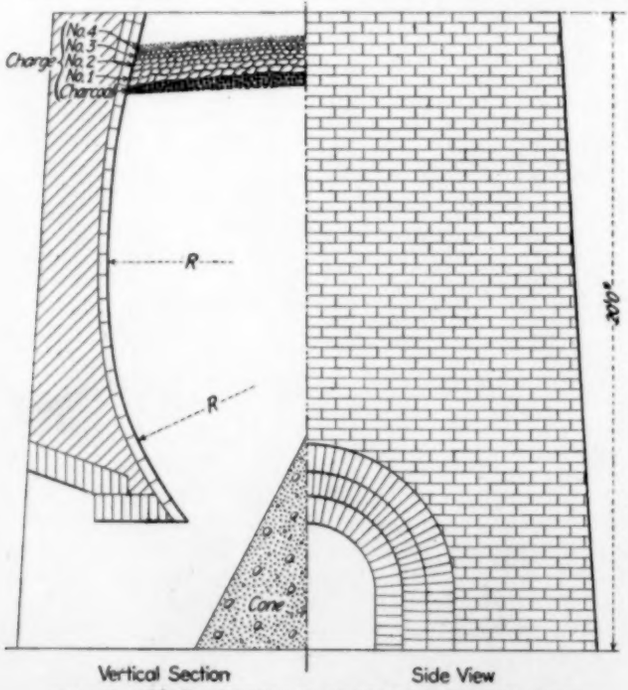
Class 1: oversize of a 3-in. grizzly; from 3 to 5 in. in diameter, averaging 4 in.

Class 2: undersize of a 3-in. grizzly, remaining on a 1-in., averaging 2 in.

Class 3: undersize of a 1-in. grizzly, remaining on ½-in. screen, averaging close to 1 in.

Class 4: undersize of ½-in. screen, averaging about ¼ in.

For satisfactory calcination in kilns, experience has shown the necessity of a certain interior shape or form to permit the charge to descend in uniform horizontal zones and to discharge promptly at the bottom. A vertical section, side view and ground plan of a kiln are shown in Figs. 1, 2 and 3. The exterior is hexagonal in form, about 16 ft. in diameter and 20 ft. high. The feed floor is level with the top. Looking down into the kiln, the interior resembles an ordinary round cistern, though less contracted at the top. The curve of the interior is determined by two radii. It allows for the expansion of the ore in calcining and produces a choking effect at the bottom above the four discharge doors. On the bottom or floor of the kiln is a central concrete cone covered with sheet iron, the apex being at a point above the arches of the four doors. This cone insures an even descent of the kiln contents and a complete removal of the finished product through the four doors.



FIGS. 1 AND 2—VERTICAL SECTION AND SIDE VIEW

Method of Charging a Kiln

The initial charging of a kiln consists in building up with wood from the kiln floor to a point above the arches of the doors. The first charge of ore is placed on the wood, consisting of successive layers of the different classes. Class 1 is spread evenly over the wood; then Class 2 is carefully fitted into the interstitial spaces on Class 1 and built up to a layer of desired thickness. Class 3 is superimposed on Class 2, and finally the fine ore, or Class 4, is added, making a level floor of gravel-like structure. Upon this foundation the regular charging begins; first broken charcoal or coal, about 4 per cent of the weight of the ore-charge, followed by layers of ore as above described, beginning with coarse and finishing with fine. In this way zone upon zone of fuel and ore is built up in the furnace, great care being taken in placing the successive layers by hand so as to form porous charges that will ignite readily after combustion is started in the wood at the bottom.

After calcination is started, the product is removed periodically at the bottom of the kiln through the four doors simultaneously, and immediately a new charge is placed in the furnace at the top. The quantity withdrawn at any one time is about equal to the quantity in a charge, and in this way the content of a kiln is kept constant and at a level about one foot below the feed floor. In a kiln 20 ft. high there will be from 8 to 10

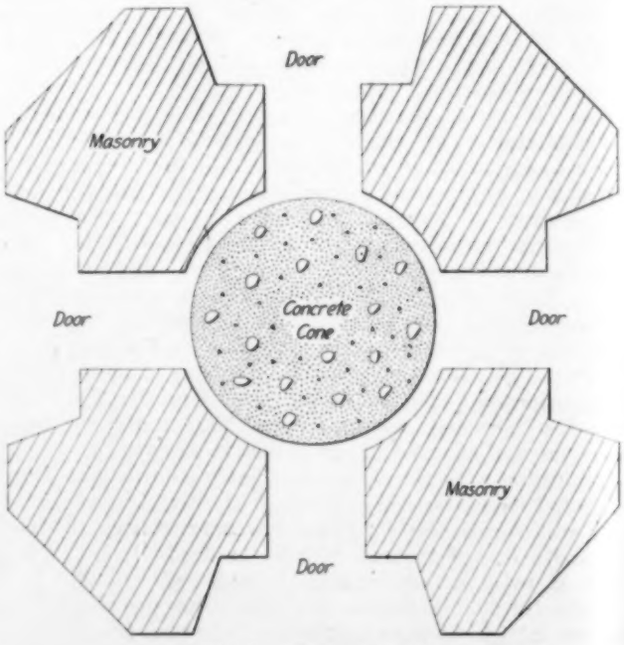


FIG. 3—GROUND PLAN

zones of the type described, each becoming ignited from the one below. Thus, after once blowing in the furnace the process continues indefinitely, the addition of a new charge coinciding with the periodic removal.

Often a kiln will run thus for months much the same as an iron or lead blast-furnace. The heat, however, is not intense, and the men engaged in placing a new charge stand in the top of the kiln without discomfort. Each kiln of the type described has a capacity of from 20 to 36 tons of product per day, depending on the nature of the ore and the proportions of the several classes or grades comprising the charge. A kiln will hold 90 tons of crude and 70 tons of calcined ore. The charges are carefully weighed, and the operation is controlled by the proportion of each class of ore charged.

Empirical Calculation of Charges

The empirical nature of the calcination process as conducted in kilns is illustrated by the method of calculating charges. An ideal charge is given in Table II. This system was developed in the pioneer work with the particular type of kiln above described. Each of the four classes of ore is given a factor in reverse order to its class number. Thus Class 1 has a factor of 4; Class 2 a factor of 3; Class 3 a factor of 2; and Class 4 a factor of 1. The unit of quantity is the wheelbarrow load, about 400 lb. Experience developed the fact that the number of barrow loads of each class could be varied, provided the total number was maintained at 28. As shown in Table II, an ideal charge gives a sum of "resultants" equal to 100; but the satisfactory working of the kiln was found to admit of such changes as would give a variation in the sum of resultants between 97 and 107. Different types of charge naturally affect the total number of charges per 24 hours. As shown in Table III, the variation of resultant from 97 to 107 gives a variation in daily tonnage from 20 to 36 for the particular type of kiln under consideration.

Rotary Kilns for Fine Ore

In preparing ore for the kiln process of calcination there results a superabundance of fine material which cannot be used in the charge, for the whole content of the furnace must be porous enough to afford proper draft. This surplus of fine ore must be calcined in a rotary inclined kiln, the charge being fed at the upper or cool end and gradually working down toward the fire-box, in front of which the calcium product is discharged. Oil or coal is used as fuel for these rotary kilns. Calcination is less complete, however, than in the vertical type. The cost for labor is much less than in the method first described, and this suggests that some mechanical method of calcination must be devised if the process is to be utilized as a means of concentration in this country, where the labor cost is high.

The question of dust losses has been raised in connection with calcination as a possible objection to the process. There is, of course, some dust loss, and precautions must be taken to avoid loss of fine material in transportation. Nevertheless the process has been found highly profitable in certain places. Another objection is that the handling of the calcine is disagreeable to laborers, particularly when there is much lime in the ore. When lime forms a substantial constituent of the ore it is not advisable to expose the calcine to the weather for a long time, as that part of the limestone which has been altered in the kiln will slake and increase in weight and consequently reduce the percentage of zinc in the calcine. The zinc, however, suffers very little in this regard, and calcines transported by rail to the Atlantic seaboard and thence by boat to Europe, often several months en route, show little alteration.

Denver, Colo.

Photochemistry

A Lecture Delivered Before the Teknik Club in Denver

BY HARRY A. CURTIS, PH.D.

Everyone has observed that light may bring about, or at least hasten, certain chemical actions. That growing plants require sunlight, that wall-paper fades, that laundered linen is whiter when dried in direct sunlight, that our hands and faces tan in the summer, these are facts of photochemistry too evident to be overlooked. To a lesser degree the photochemical processes of photography are known to the layman, but the modern science of photochemistry is known only to a few investigators. It may, therefore, be of some interest to the readers of this journal to consider the broad outlines of a branch of chemistry which offers great promise of becoming vastly important to man.

It is not the purpose of the present paper to trace the development of the science of photochemistry, but rather to summarize the results which research in this field has yielded and to point out certain large problems which are now occupying most attention.

To discuss a highly specialized branch of any sicence in language which shall not be so technical as to bewilder the average reader of a scientific journal, and yet not lose that precision which a special nomenclature brings to any science, is no easy task. The attempt will be made in this article to steer a happy middle course between the inaccuracies which inhere in a popular discussion of a technical subject and the formidable language behind which every science finally takes refuge from popular attack.

The term photochemistry has unfortunately become associated with the better known word photography. It must therefore be pointed out here that photography is but one branch of a very much wider science, photochemistry. Photochemistry has been defined as the study of the effect of light on chemical reactions, the term light being used in its broader meaning to include the infra-red and ultraviolet regions of the spectrum.

Visible light includes waves of radiant energy whose lengths* lie roughly between 800 $\mu\mu$ and 400 $\mu\mu$. This range is but a very short piece of the total spectrum of radiant energy, which is now known to extend from waves less than a millionth of a millimeter long up to those having a length of several hundred miles. It is instructive to give in Table I the various parts of this spectrum which have been investigated to date.

TABLE I—SPECTRUM OF RADIANT ENERGY

Wave-Length	Name of Wave	Investigator
0.01$\mu\mu$ to 1$\mu\mu$	Röntgen or X-rays	Röntgen, 1895
1$\mu\mu$ to 100$\mu\mu$	Uninvestigated	
100$\mu\mu$	Shortest ultraviolet	Schumann, 1893
200$\mu\mu$	Ultraviolet rays	Cornu, 1878
200$\mu\mu$ to 400$\mu\mu$	Ultraviolet rays	Ritter, 1801
400$\mu\mu$ to 800$\mu\mu$	Visible rays	Newton, 1666
800$\mu\mu$ to 1μ	Ultrared rays	Herschel, 1800
1μ to 10μ	Heat rays	Langley, 1886
10μ to 61.4μ	Heat rays	Rubens, 1896
108μ	Heat rays	Rubens, 1910
218μ and 343μ	Heat rays	Rubens, 1911
343μ to 1 mm	Uninvestigated	
4 mm	Shortest electrical waves	Lampa, 1895
6 mm	Electrical waves	Lebedew, 1895
1 cm. to 10 m	Electrical waves	Hertz, 1889 Lodge, 1890 Righi, 1894
10 m. to 1 km	Electrical waves	Feddersen, 1892
1 km. to 100 km	Electrical waves	Tesla
100 km. to 1000 km	Electrical waves	Lodge, 1888

It will be seen in Table I that if the visible portion of the spectrum be called unity, then the total range represented in the table will be 2.5×10^{14}. To get an idea of the magnitude of this ratio, suppose the length of

*1μ = 0.001 millimeter;
1$\mu\mu$ = 0.000001 millimeter.

the visible spectrum to be a meter. Then the total length represented would span the distance from earth to sun about 170 times. It is probable that radiant energy of every wave length will be found to have an effect on chemical reactions. As yet, however, research in this line has been almost completely limited to the visible and ultraviolet parts of the spectrum, to heat rays, and, in very recent years, to a few scattered investigations in the region of very short waves.

Available Sources of Light

Our greatest source of light is, of course, the sun. Professor Luther estimates that the sun delivers continuously to the earth about 200,000,000,000 hp., which is about 1,000,000 times more than is generated by all the engines on earth. Of this enormous quantity of energy, plants utilize a small portion, part is reflected away from the earth, and the balance is dissipated as heat. To store up and utilize a larger part of this energy with which the sun floods the earth is one of the dreams of photochemists.

While it is probable that any successful commercial process of photochemistry must utilize sunlight, it has been found that sunlight is a most unsatisfactory source of light for purposes of photochemical research. The difficulty is, of course, that the intensity of the sunlight is never constant for more than a few minutes at a time. Aside from this, the earth's atmosphere filters out the shorter waves of light, the very ones which are the most effective in most photochemical processes.

Of the various artificial sources of light, only a few are of sufficiently high intensity to be used in photochemical studies. The electric arc has sufficient intensity, but is difficult to keep burning steadily. The same difficulty is met with in the case of the electric spark, although several successful pieces of research have been accomplished using the spark as a light source. The Nernst glower burns steadily, and gives a continuous spectrum, but the light is not rich in the short waves. By far the most successful lamp for purposes of photochemical research are the mercury-vapor lamps. These are easy to operate, and give an intense light which is very rich in short waves. The spectrum of the light emitted is, of course, discontinuous. This is an advantage where it is desired to carry out research using monochromatic light, for if the light be spread out with a prism, any particular line may be used as a light source by simply screening off the rest of the spectrum. Where all the light is to be used, the spectrum may be enriched by replacing the mercury with an amalgam.

If the shorter waves of the mercury-vapor light are to be utilized, however, the lamp may not be constructed of ordinary glass, for this is not transparent to waves shorter than about 350 $\mu\mu$. Many attempts have been made to prepare glass which should be more transparent to the shorter waves. These attempts have been successful to the extent that glass has been produced at the Jena works having a transparency extending to nearly 250 $\mu\mu$. This, however, only partially solves the problem. There are indeed very few materials which do possess the desired properties. The lower limits of transparency of a number of common materials are given in Table II.

Of these materials, quartz is the one best adapted, and since the methods of working quartz have been developed, this material has found wide use in photochemical research, both in the manufacture of lamps and of all sorts of reaction vessels.

The Measuring of Light

The problem of measuring light in photochemical research is one which differs radically from the measuring of light for ordinary illuminating purposes. In the latter it is the visual result which is in question, and therefore any of the visual photometers may be applied. In photochemistry, however, the problem is usually to determine the rate at which radiant energy is being delivered to the reaction vessel. This at once eliminates all visual photometers, for the human eye is not equally sensitive to light waves of all lengths. It is most sensitive to yellow-green light, and is, of course, blind to the ultraviolet which is so active photochemically. Many chemical photometers, or actinometers as they are called, have been invented, but all of them have the same fault in that they are not equally sensitive to light of all wave-lengths. Ignorance of this fact has rendered of little value much of the early work in photochemistry.

TABLE II—THE LOWER LIMITS OF TRANSPARENCY

FLINT GLASS	One centimeter thickness transmits from 50 to 80 per cent at 350$\mu\mu$. Less than 1 per cent is transmitted at 305.
UVIOL GLASS	Lower limit at 253$\mu\mu$.
FLUORITE	One centimeter thickness transmits 83 per cent at 186$\mu\mu$ and the transparency is still high at 100$\mu\mu$
ROCK SALT	Transparency is high at 186$\mu\mu$
QUARTZ	Transmits 94.2 per cent at 222$\mu\mu$ and 67.2 per cent 186$\mu\mu$; at beyond 186$\mu\mu$ the transparency falls off rapidly.
CALCITE	Less transparent than quarts for short waves.
MICA	Strongly absorbs the short waves.
WATER	Transparency is high at 193$\mu\mu$. At 186$\mu\mu$ it is less transparent than is quarts.
GLYCERINE	Transparency is high at 240$\mu\mu$.
CANADA BALSAM	Transparency is high at 330$\mu\mu$.
GELATINE	Transparent down to 257$\mu\mu$.
AIR	Transparency is high at 194$\mu\mu$; at 186$\mu\mu$ the absorption is very high, and no waves shorter than 165$\mu\mu$ are transmitted.

One of the instruments most used for measuring light intensity in modern photochemical research is the thermopile. This instrument is based on the well-known fact that when radiant energy falls upon a junction between two different metals, that part of the energy which is absorbed heats the junction and will cause a current of electricity to flow from the one metal to the other through an external circuit. If a delicate galvanometer be included in the circuit, the current may be measured, and by comparing this current with that produced by a radiation of known intensity, the total radiation from any source may be measured. The radiation of known intensity is obtained from a so-called "black-body" radiator. Stefan discovered empirically, and Boltzmann derived by thermodynamic reasoning the law now known as the Stefan-Boltzmann law, which states that the total radiation from a perfectly black body is proportional to the fourth power of the absolute temperature. Stated algebraically $S = CT^4$, where S is the total radiation, T the absolute temperature, and C a natural constant having a value approximating 1.71×10^{-5} ergs per square centimeter per second. It is, of course, not possible to construct a "perfectly black body," in the sense here used, but the deviation of certain actual radiators from the ideal is negligible.

Another instrument often used to measure radiant energy is the radiomicrometer. In this instrument a thermopile and small circuit are suspended by a fine fiber between the poles of a magnet. When light falls upon the thermopile, a small electrical current flows through the circuit and the suspended system is turned by the magnetic field as in the D'Arsonval galvanometer, the effect being magnified by an optical lever.

Types of Photochemical Reactions

In general it may be said that light is capable of bringing about every type of chemical reaction. The number of photochemical reactions now known is far too large to allow a consideration of each one individually. We shall, therefore, only point out a few examples under each type of reaction.

Oxidation Reactions.

a. Lead sulphide is oxidized to lead sulphate by light and air:

$$PbS + 2O_2 = PbSO_4$$

b. Sulphur dioxide is partially oxidized to sulphur trioxide by light and oxygen:

$$2SO_2 + O_2 \rightleftarrows 2SO_3$$

Reduction Reactions.

Silver salts, iron salts, mercury salts, bismuth salts, etc., are reduced by light, the metal being liberated in some cases, in others a salt corresponding to a lower valence of the metal being formed. The ordinary process of blue-printing involves the photoreduction of an iron salt.

Polymerization.

a. Under the influence of light oxygen is converted partially into ozone, the equilibrium between the oxygen and the ozone depending upon the temperature and the intensity of the light used.

$$3O_2 \rightleftarrows 2O_3$$

b. In organic chemistry a great many polymerizations have been accomplished by means of light. As a simple example we may cite the conversion of cyanogen into paracyanogen:

$$x(CN)_2 = (CN)_{2x}$$

Depolymerization.

Ozone is partially decomposed into oxygen, the decomposition proceeding until the equilibrium point has been reached as in the formation of ozone from oxygen.

Allotropic Changes of the Elements.

Under the influence of light sulphur which is soluble in carbon disulphide, so-called $S\lambda$, is converted into sulphur which is insoluble in carbon disulphide, so-called $S\mu$.

Likewise yellow phosphorus is converted into the red modification at a very much higher rate than occurs in the dark.

Isomeric Changes.

Light is especially active in bringing about isomeric changes in organic chemistry. For example, maleic acid becomes fumaric acid under the influence of light, ortho-nitrobenzaldehyde becomes ortho-nitrobenzoic acid, isocrotonic acid becomes normal crotonic acid, cumaric acid becomes cumarin, etc.

Light may also bring about stereo-isomeric changes, as, for example, in nearly all the fulgides, where the change is usually accompanied by a change in color.

Hydrolysis.

Hydrolyses, especially in organic chemistry, are readily accomplished by light. Thus acetone and water react to form acetic acid and methane:

$$CH_3COCH_3 + H_2O = CH_3COOH + CH_4$$

Ethyl bromide yields alcohol and hydrobromic acid:

$$C_2H_5Br + H_2O = C_2H_5OH + HBr$$

Synthesis.

a. Hydrogen and chlorine combine to form hydrogen chloride with explosive rapidity in direct sunlight:

$$H_2 + Cl_2 = 2HCl$$

b. Carbon monoxide and chlorine combine to form phosgene:

$$CO + Cl_2 = COCl_2$$

c. In organic chemistry a great many syntheses have been carried out by means of light, this method often affording a direct and simple means of carrying out reactions which may be accomplished only with much difficulty by other means.

It has long been known that the halogenation of organic compounds is much hastened by exposing the reaction mixture to sunlight. In this connection it is interesting to note that a close parallelism exists between halogenation at high temperatures and halogenation in sunlight.

Decompositions.

a. Light decomposes hydrogen iodide very readily in the presence of oxygen or air:

$$HI = H + I$$

b. Hydrogen peroxide is decomposed into water and oxygen:

$$H_2O_2 = H_2O + O$$

The rate of decomposition is greatly influenced by the presence of a trace of foreign substance in the solution, in most cases the solution becoming more stable upon the addition of the foreign substance.

c. Under the influence of light of short wave-length, most organic compounds are decomposed into simpler substances. In Table III are given the decomposition products of a number of simple organic compounds.

TABLE III—DECOMPOSITION PRODUCTS OF ORGANIC COMPOUNDS

Compound	PERCENTAGE OF PRODUCTS					
	CO_2	CO	H_2	CH_4	C_2H_6	C_4H_{10}
Methyl alcohol	5	8	87	..	..	..
Ethyl alcohol	..	22	63	..	15	..
Propyl alcohol	..	16	69	..	..	15
Formaldehyde	9	39	46	6	..	..
Acetaldehyde	5	39	33	..	23	..
Propyl aldehyde	6	37	37	..	..	20
Formic acid	59	21	19	1	..	..
Acetic acid	41	14	13	13	19	..
Propionic acid	41	15	15	15	..	14
Acetone	..	49	..	5	46	..
Acetone and water	1	44	..	9	46	..

A Few Unsolved Problems of Photochemistry

I. THE CHLOROPHYLL PROBLEM

Chemical action may easily be caused to produce light; indeed artificial light was produced in no other way until modern times. To drive chemical actions by means of light is, however, a process about which but little is known, although it is a process which every growing plant successfully accomplishes.

The photochemistry of plant assimilation is now receiving much attention. In this process, carbon dioxide and water are the raw materials, carbohydrates the chief of the finished products. Between these two systems, carbon dioxide plus water on the one hand, and carbohydrates on the other, is a big gap in energy which the plant crosses by utilizing the energy of sunlight. The active agent in this transfer, the chief engineer if you please, is a certain green substance called chlorophyll, but beyond the fact that the goods are delivered on the proper side of the gap, not much is known about the process.

It is possible to manufacture carbohydrates from water and carbon dioxide in the laboratory. It is even possible to do this by means of light without the aid of chlorophyll. Here, however, an unfair advantage must be taken by using light of shorter waves than occur in sunlight. It seems probable that chlorophyll plays the part of a photocatalyst, by the aid of which plant assimilation may proceed in light of much lower waves than could otherwise be utilized. There are other examples of such photocatalysis or "optical sensitization," of which a familiar one is the use of certain dyes to extend the sensitiveness of the photographic plate into the yellow and red region of the spectrum.

In recent years much progress has been made by Professor Willstätter and his students toward solving the "chlorophyll problem" but much remains unknown about this photochemical process to which all the coal beds on earth owe their origin. Could we but learn to "speed up" the process of plant assimilation, as we can speed

up the process of combustion, sun-power would certainly become a factor in the industrial world.

II. THE LAWS OF PHOTOCHEMISTRY

A study of the kinetics of photochemical reactions is complicated by a number of troublesome factors, and progress in this line has been slow.

In order to be photochemically active, light must be absorbed. The fact of light absorption by a system, however, does not necessitate chemical action, since the light absorbed may be converted into heat. When photochemical action does occur, the degree of absorption is an important factor in determining the mechanics of the reaction. It appears that in reversible reactions with a large absorption of light, the velocity of the reaction becomes simply proportional to the amount of light absorbed per unit time, which is obviously a law analogous to the Faraday law of electrolysis. With an irreversible reaction having a small absorption, however, the mass law may be applied with a few modifications.

But little is known about the catalysis of photochemical reactions, which is not surprising in view of our almost complete ignorance as to the nature of catalysis in general.

The effect of temperature on photochemical reactions is much smaller than on other reactions. An increase of temperature of 10 deg. will usually double or treble the rate of an ordinary reaction, while the same increase of temperature seldom multiplies the rate of a photochemical reaction by more than one and a half, and in many cases has very little effect at all. It is well known, for example, that a photographic plate is nearly as sensitive at the temperature of liquid air as it is at ordinary temperatures.

The whole subject of photochemical kinetics is, as yet, much confused, and a great deal of quantitative experimental data are needed. There is undoubtedly some relationship between the chemical reactivity of a substance under influence of light and its molecular structure, just as there must be between its color, itself a photochemical phenomenon, and its molecular structure.

III. COLD LIGHT

The total energy radiated by a body is proportional to the fourth power of its absolute temperature (Stefan-Boltzmann law). This energy, however, is not evenly distributed in the spectrum, but for each temperature there is a characteristic distribution curve having a maximum at a certain wave length. As the temperature is raised, this maximum shifts toward the region of shorter waves. We are all aware of the qualitative operation of this law, that when the temperature of a body is gradually raised, heat waves are at first emitted, and later light waves of the longest wave length visible, i.e., red light. Obviously all "perfectly black bodies," i.e. such as follow the Stefan-Boltzmann law, would grow "red hot" at one definite temperature, while actual bodies, which differ somewhat from the ideal black body, will become red hot at temperatures which differ somewhat, but are not far from that of the ideal black body. All incandescent lamps operate on this principle, and the radiations produced are by no means limited to the visible regions of the spectrum, so that a considerable portion of the electrical or chemical energy used in the operation is wasted so far as the production of light is concerned.

So much for the ordinary emission of light. There are many cases known, however, where light is emitted from a body whose general temperature is far below red heat. Several kinds of luminescent bacteria are known; the fire-fly and glow-worm are common; phosphorus glows when exposed to air and moisture; fluorite emits a bluish-green light when gently warmed; certain sulphides will glow in the dark after a preliminary exposure to light; many chemical reactions are accompanied by the production of light at ordinary temperatures; crystallization is often accompanied by tiny flashes of light, and a similar phenomenon is to be observed when certain crystals are crushed.

In all these cases the light is produced at temperatures which are several hundreds of degrees below that which would be required by the ordinary laws of radiation. This phenomenon, the production of "cold light" by any of the above-named means, goes under the general name of luminescence. The mechanism of the process or processes is unknown.

That all forms of energy are mutually and quantitatively interconvertible is one of the axioms of physical science. The practical problem of converting one form into another is, however, a problem of gigantic proportions, one which has claimed the attention of men from earliest times. That the problem has in part been solved, is evidenced by the tremendous operations of modern industry. It is perhaps not too much to say that modern civilization could not have been attained had man not learned to transmute energy. There is, however, little room for pride in the achievement, for at every transfer much of the valuable free energy is spilled into the great dead-level ocean of heat to which every form of energy seems destined to revert ultimately. Engineers talk much of "efficiency" and seem to feel that the word lends a luster of achievement to their discourse. The real fact is that the word "efficiency" hides the chagrin with which they deliver at its destination so small a fraction of the total amount of energy which they bravely essayed to transform.

It so happens that some of the transformations are much easier to make than others; mechanical energy may be transformed into electrical energy, and vice versa, with but little loss, while in the transformation of heat energy into mechanical energy but little is saved. Among those transformations in which least skill is shown is the production of light. If produced through the chemical process of combustion, but a small part of the chemical energy liberated becomes light; the various electric incandescent lamps do better, but even here a considerable portion of the electrical energy is spent in producing waves of radiant energy far too long to be seen. To transmute chemical energy wholly into visible radiant energy is a problem which has so far utterly defied practical solution. The photochemist is still struggling with the problem. If therefore he is discovered hovering about some swampy place with a fine-mesh butterfly net, be sure that he is only pursuing common fire-bugs, although the results so far achieved in this line would indicate that he has captured only will-o'-the-wisps.

Coal and Coke in Canada.—The production of coal and coke in Canada during 1914 is the subject of an advance chapter of the annual report of the mineral production of Canada by the Department of Mines.

Removal of Logwood Embargo in Jamaica.—According to a telegram from the American Consulate at Kingston to the Department of Commerce, the embargo on logwood, logwood chips and logwood extract, including hematin crystals and other logwood preparations, has been raised.

The copper-nickel mattes produced by the Canadian Canadian Copper Co. and the Mond Nickel Co. in Canada are shipped respectively to New Jersey and to Wales for refining. Gold, silver, platinum and palladium are recovered as by-products from this matte.

Refrigeration in France*

BY L. MARCHIS

I—The Technique of Very Low Temperatures

The industrial production of liquid air, of oxygen, of nitrogen and of hydrogen, have, thanks to the labors of G. Claude, made great progress during the last few years. We shall here give in résumé the more important results of his researches.

1. PRODUCTION OF LIQUID AIR

Claude utilizes the external work produced by the passage of air from a high to a lower pressure, but he applies this well-known method in the following manner for the production of low temperatures.

The air compressed to a maximum pressure of 40 atmospheres, passes into a cooler where its temperature is reduced as we shall see at a later point.

This current of air, cooled and compressed to 40 atmospheres, is then to be passed into a cylinder where work is produced by expansive action on a piston. But before reaching this expansion cylinder, a part of the air is diverted from the main conduit. This diverted air, at a pressure of 40 atmospheres, reaches a cooled inclosure (liquefier) where the air is liquefied at the temperature of — 140 deg. C. The remainder of the air (from which the portion has been diverted) then passes on to undergo an expansion by stages in a series of cylinders within which are pistons on which the air acts with the production of work.

The air, exhausted after the first expansion, is circulated about a first liquefier traversed by the diverted air (referred to above) at 40 atmospheres pressure. This air, which has undergone a first expansion, is warmed and then expanded in a second auxiliary cylinder. The air of the second expansion is sent into a second liquefier similar to the first.

Finally this air, twice expanded, is caused to circulate in the cooler where it operates as a cooling agent for the air compressed to 40 atmospheres which comes from the compressor.

For machines using a compressor of 75 hp., the delivery of liquid air is equal to 0.85 liter (0.224 gal.) per effective horsepower-hour.

2. SEPARATION OF THE OXYGEN AND NITROGEN OF THE AIR

Liquid air is especially important because it tends to become the only industrial source of oxygen and of nitrogen. The production, at a low price, of these two gases is a problem, the solution of which is of great importance for metallurgy and for the fertilizer industry. How may these two gases be derived from liquid air? This problem we shall now proceed to examine.

Oxygen and nitrogen are two substances of which the critical points are distinctly different, as follows:

— 119 deg. C. and 51 atmospheres for oxygen.
— 146 deg. C. and 34 atmospheres for nitrogen.

If, in the plane *POT* (*OP* vapor pressure ordinates: *OT* temperature abscissæ) we trace the curve of vapor tension for the saturated vapors of oxygen and of nitrogen, it will be found that the curve for nitrogen will lie above that for oxygen. At the same temperature, lying below the critical temperature, nitrogen is liquefied under a pressure higher than oxygen; at the same pressure, inferior to the critical pressure of the two gases, nitrogen is liquefied at a temperature lower than oxygen. Thus under atmospheric pressure oxygen is liquefied at — 182.6 deg. C. and nitrogen at — 194.4 deg. C.

*A paper read before the International Engineering Congress in San Francisco.

3. SPECIAL CIRCUMSTANCES PRESENTED BY THE LIQUEFACTION OF AIR (MIXTURE OF OXYGEN AND NITROGEN)

The liquefaction of air, that is of the mixture of the two gases oxygen and nitrogen, presents certain special conditions which we must here note.

If, at a constant temperature T, sufficiently low, air is compressed in a closed chamber, the following phenomena are observed:

(1) At a certain value P_0 of the pressure, there appears the first drop of liquid; we shall designate the state of equilibrium characterized by the value P_0T by the term "dew-point."* For all pressures less than P_0 the air is in the state of homogeneous vapor; from P_0 upward, separation of the air in two phases is produced.

(2) If, at the constant temperature T, the volume of the air is continuously decreased, *the pressure rises;* at the same time the amount of the liquid phase increases and the amount of the vapor phase decreases. If the increase in pressure is continued, the air passes entirely into the liquid state; the values P_1 and T of the pressure and of the temperature at this instant characterize what we shall term the "boiling point."

For all pressures exceeding P_1 the air is in the state of homogeneous liquid; for all pressures comprised between P_0 and P_1 the air is divided into two phases, one liquid and one vapor.

(3) The "dew-points" and the "boiling-points," obtained at diverse temperatures trace in the plane *POT* (*OP*, ordinates; *OT*, abscissæ) on the one hand the *line of dew* and on the other the *line of ebullition* of the gaseous mixture considered.

The line of ebullition lies entirely above the line of dew; the two lines come together at the critical point, that is, in the present case, at the point for which the co-ordinates are 39 atmospheres and — 140 deg. C.

(4) For each system of values (*TP*) of the temperature and of the pressure (*T* below the critical temperature and *P* lying between P_0 and P_1), there exists a state of equilibrium of the combination of the two phases in contact one with the other. In this state *the compositions of the two phases are different.*

(5) Let us apply the term "oxygen quality" of the phases, liquid and gaseous, to the percentage of oxygen (the more liquefiable element) in each of them.

The oxygen quality of the liquid phase is, in the state of equilibrium, always higher than that of the gaseous phase. The liquid is always richer in oxygen than the vapor.

When, *at constant temperature the pressure rises,* the oxygen qualities of the phases, liquid and gaseous, continuously decrease to the complete liquefaction of the mixture.

4. THE SEPARATION OF OXYGEN AND NITROGEN CAN NOT BE REALIZED WHEN, IN THE LIQUEFACTION OF AIR, THE LIQUID AND GASEOUS PHASES ARE MAINTAINED IN CONTACT

Thus when air is liquefied (volumetric content 21 per cent of oxygen) the first drop of liquid which is formed contains both oxygen and nitrogen and its "oxygen quality" is 47 per cent. The values of the "oxygen quality" continuously decrease in proportion as the volume of the liquid phase increases: It is thus that liquid at 34 per cent can only be in equilibrium with vapor at 12.5 per cent. But, so long as a gaseous phase exists, its oxygen quality is very different from 0; it remains greater than 7 per cent.

If the two phases, liquid and gaseous, obtained by the progressive liquefaction of air are maintained in con-

*Note that this is entirely distinct from the dewpoint relating to water vapor and air. (Tr.)

tact, it is impossible to prepare gaseous nitrogen free of oxygen, that is with a sufficient degree of purity.

5. DISTILLATION OF LIQUID AIR UNDER CONSTANT PRESSURE

It is not the same, however, when, maintaining the pressure constant, the liquid phase is removed as rapidly as formed. In such case the phenomenon encountered is the inverse of that met with when liquid air is distilled under constant pressure. In this case the oxygen qualities of the phases, liquid and gaseous, continuously increase; these phases, separated one from the other as rapidly as formed, tend toward a content of pure oxygen; at the same time the temperature of ebullition rises from a value nearly that of pure nitrogen to the temperature of ebullition of pure oxygen.

6. CONDENSATION OF AIR UNDER CONSTANT PRESSURE WITH ELIMINATION OF THE LIQUID PHASE

Inversely, if, under constant pressure, air is progressively condensed, removing the liquid phase as rapidly as formed, liquids and gaseous residues are obtained less and less rich in oxygen; at the same time the temperature of condensation drops and tends toward the temperature of ebullition of pure nitrogen under the pressure considered. There is thus obtained a gaseous mixture richer in nitrogen than that obtained by leaving the liquid and gaseous phases in contact.

However, in order to obtain a gaseous residue sensibly free of oxygen, it is necessary in this method to liquefy practically all of the air.

7. THE COUNTER-FLOW METHOD OF G. CLAUDE

A better result is realized by the use of the method designed by Mr. Claude under the name of regenerative or "counter-flow" method.

Let us assume in any case the progressive condensation of the air and the removal of the liquid phase as rapidly as formed. But assume further that the liquid thus obtained meets, in separating from the gas, a gaseous mass richer in oxygen. The liquid is colder, as a consequence of the high percentage of nitrogen which it contains; a part of the more condensable oxygen of the gaseous mixture will then become liquid and will take the place of the nitrogen which is vaporized. Thus by a counter-flow circulation of the liquid and the gas of different oxygen qualities, there is obtained, on the one hand, a liquid very rich in oxygen, and on the other, *gaseous nitrogen practically pure*. The latter comes, furthermore, in large part from air treated but not liquefied.

Mr. Claude has realized as follows the principle of the "counter-flow":

A form of small tubular boiler is disposed in such manner that its axis is vertical. The tubes are surrounded on the outside by the liquid air. There is then passed into the inside at the bottom a current of cold compressed air. This in passing up through the tubes is progressively liquefied, producing both liquids and vapors less and less rich in oxygen. The vapors continue to rise toward the top of the tubes while the condensed liquid descends toward the bottom. This liquid, in falling into the lower receptacle, meets the gaseous portions rich in oxygen, and thus is produced the continuous exhaustion, of which the principle has been set forth above. There is thus liberated from the top of the nest of tubes practically pure nitrogen, while liquid specially rich in oxygen is continuously drawn from the bottom.

8. THE PREPARATION OF PURE OXYGEN AND OF NITROGEN 97 PER CENT OR 98 PER CENT PURE

A second stage remains to be realized. Instead of air with an oxygen content (volumetric) of 47 per cent., it is required to obtain oxygen practically pure. This is realized by means of methods of *rectification* based on those employed in the manufacture of alcohol.

In a vertical column there circulate: (1) From bottom to top, oxygen vapor practically pure; (2) from top to bottom, liquid with a high percentage of nitrogen. The latter, colder, condenses the oxygen and allows its nitrogen to escape in the gaseous condition, according to the procedure above developed in the description of the counter-flow method.

The apparatus is composed of a sort of tubular boiler with axis vertical. This is extended on top by a sheet-metal cylinder of the type employed in the distillation of alcohol. The vertical tubes of the boiler are surrounded by liquid oxygen practically pure. On the inside and at the base of the tubes, air is led in under a pressure of 5 atmospheres. As has been explained above, this air is liquefied, giving a liquid air with excess of oxygen in the lower part of the tubes and nearly pure gaseous nitrogen in the upper part. The latter is carried through the liquid oxygen and is liquefied in turn. In consequence of its pressure, the liquid with excess of oxygen flows out continuously at the middle part of the rectifying column, while the liquid nitrogen, very poor in oxygen, overflows at the top of the column.

Furthermore, the oxygen which surrounds the boiler is vaporized, as a result of the disengagement of heat resulting from the condensation of the air in the interior of the tubes. This vapor of oxygen, in rising through the column, encounters liquids richer and richer in nitrogen. It is then condensed, taking the place of the nitrogen which is vaporized. Thus practically pure liquid oxygen falls back into the boiler, while pure nitrogen is liberated at the upper part of the rectifying column. The mass of liquid oxygen which falls back into the boiler is superior to that which is vaporized in rising through the column. The excess of oxygen is evacuated and carried through a cooler to the gasometer or point of utilization. This cooler serves to cool the air at 5 atmospheres pressure which is introduced into the base of the apparatus.

The ensemble of this equipment (cooler, liquefier, rectifying column) is surrounded by a thick layer of heat insulating material which opposes the inflow of heat.

The delivery of this apparatus is approximately 1 cu. m. (35.3 cu. ft.) of pure oxygen per effective horse-power-hour for the size producing 50 cu. m. (1765.7 cu. ft.) per hour; 1.2 cu. m. (42.4 cu. ft.) for the sizes producing 100 to 200 cu. m. (3530-7063 cu. ft.) per hour; 1.5 cu. ft. (53 cu. ft.) for the sizes producing 1000 cu. m. (35,300 cu. ft.) per hour, operating at pressures of 10 atmospheres at the most. For these latter, aside from the motive power, the costs (labor and fixed charges) will not exceed 0.01 franc per cubic meter of oxygen (0.005 cent per cubic foot). With the motive power available in the neighborhood of furnaces and waterfalls, the total cost of oxygen will result, according to Mr. Claude, about 0.02 franc per cubic meter (0.01 cent per cubic foot) or 15 francs per ton (density of liquid oxygen 1.33) ($2.62 per ton of 2000 lb.)

At the present time the forty plants which are operating with the preceding apparatus (Liquid Air Company) are producing annually 24,000,000 cu. m. (847,000,000 cu. ft.) of liquid oxygen. Operating at the same time, they would furnish 3000 cu. m. (105,900 cu. ft.) of oxygen per hour while liquefying 40 tons of air per hour. The factory at Boulogne-sur-Seine operates with two units each of 150 cu. m. (5300 cu. ft.) capacity.

9. THE PREPARATION OF NITROGEN WITH 0.2 PER CENT OXYGEN

Nitrogen with 2 per cent or 3 per cent of oxygen, the preparation of which we have just described, cannot be employed in the manufacture of cyanamide.

In improving the method of rectification described above Mr. Claude has succeeded in preparing nitrogen containing not more than 0.2 per cent of oxygen (called nitrogen 99.8 per cent pure).

The principal fault of the apparatus for the production of pure oxygen consists in its insufficient production of liquid nitrogen. The residual gases coming from the progressive condensation of the air do not find, in passing again through the bath of oxygen, a medium sufficiently cold to completely condense them, but in the rectifying column there may be found a part sufficiently cold to secure the more complete liquefaction of the nitrogen resulting from the progressive liquefaction of the air.

It is for this reason that in the new equipment constructed by Mr. Claude the residual gases coming from the progressive liquefaction of the air, instead of passing into the liquid oxygen, are conducted through a serpentine liquefier placed in a sufficiently cold region of the rectifying column. Nitrogen, liquefied in this region, is led to the upper part of the rectifying column and flows from above downward, meeting the vapors of oxygen coming from the bath which surrounds the lower liquefier.

At the present time the Liquid Air Company, which is exploiting the processes of Mr. Claude, has in operation or under construction 18 units for pure nitrogen, each of 500 cu. m. (17,650 cu. ft.) capacity per hour. When they are all in operation at the same time, 3 tons of nitrogen per hour may be delivered.

10. THE PREPARATION OF HYDROGEN

Hydrogen is extracted from water gas. It is known that this gas contains on the average 50 per cent of hydrogen and 40 per cent of carbon monoxide. The separation of these gases is very readily made by the methods of Mr. Claude. However, the process has not as yet been carried to the point of industrial production. The Liquid Air Company is making experiments in this direction and is liquefying 150 to 200 kg. (330 to 440 lb.) of carbon monoxide per hour. This substance may be employed mixed with oxygen to form an explosive mixture available as fuel in an internal combustion engine.

11. THE EXTRACTION OF THE RARE GASES IN THE ATMOSPHERE

The counter-flow method of Mr. Claude has made possible the extraction from atmospheric air of the rare gases, helium, neon, argon, krypton and xenon. It secures, in fact, the possibility of gathering, as a by-product of the industrial manufacture of oxygen and of nitrogen, a mixture of nitrogen and at the least 50 per cent of rare gases, and more particularly of neon and helium.

To this end, the gaseous residues which are the most difficult of liquefaction are led, in the proportion of 6000 liters (1585 gal.) per hour for a delivery of air of 3500 cu. m. (123,500 cu. ft.), to the lower part of a tubular system cooled by liquid nitrogen. Under the simultaneous effect of the pressure of 4 atmospheres and of the very low temperature, the condensation of all liquefiable substances results.

The gaseous residue is constituted, if the delivery is properly regulated, of a mixture almost pure of neon and helium. An apparatus of a capacity of 50 cu. m. (1765 cu. ft.) oxygen per hour produces per day 100 liters (26.4 gal.) of neon or 30 cu. m. (1059.4 cu. ft.) per year.

Neon is now employed in tubes which light up with a rose color as a result of the passage of an electric current at high voltage. These lamps give approximately 200 cp. per running meter of tube and require about 0.6 watt-hour per candle power. By the introduction of a little mercury into the neon tubes, a beautiful blue light is obtained.

12. LIQUID OXYGEN EMPLOYED AS AN EXPLOSIVE

Liquid oxygen may serve as an explosive. Cartridges composed of ordinary lampblack, dipped for a few moments in a bucket filled with liquid oxygen and primed with a fulminate cap, constitute a formidable explosive, as powerful as gum dynamite and costing only one-fourth to one-fifth as much as the latter.

13. PRODUCTION OF TEMPERATURES DOWN TO — 211 DEG. C. BY THE USE OF LIQUID NITROGEN

Hydrogen boils under atmospheric pressure at a temperature of — 252.6 deg., but this substance is still difficult to procure in considerable quantity. Mr. Claude has indicated a simple method for obtaining in a few minutes a temperature of — 211 deg. when liquid nitrogen is available. This process is more convenient, more rapid, and permits of operating with a restricted equipment, on quantities of liquid nitrogen greater than by means of evaporation under partial vacuum.

When there is passed into a liquefied gas a rapid current of air, the liquid is cooled very much below its normal point of ebullition.

The liquid nitrogen is placed in a Dewar-d'Arsonval tube freely open to the air. There is first passed in a rapid gaseous current of hydrogen, first cooled in liquid nitrogen. In the experiment, the rapidity of the current of hydrogen is gradually raised from 20 to 25 liters (5.3 to 6.3 gal.) per minute to 50 or 60 liters (13.2 to 15.85 gal.) per minute.

Under these conditions, a period of about twelve minutes is sufficient in order to lower the temperature of the liquid nitrogen to — 210 deg. C. From this point on, the temperature remains sensibly fixed. After twenty minutes, a temperature — 211 deg. is reached; a limiting temperature corresponding to the slow congelation of nitrogen. There may be thus realized a fixed point in a manner quite as convenient as those of the boiling points of oxygen and of nitrogen.

One might employ the same general method with liquid oxygen, but in this case the limiting lower temperature would be — 204 deg. In order to obtain with this substance the same temperature by a reduction of pressure, it would be necessary to boil it under 50 mm. (1.9685 in.) of mercury.

14. RECOVERY OF VOLATILE LIQUIDS. METHOD OF MR. CLAUDE

In several important industries (artificial silk of Chardonnet; smokeless powder; celluloid, etc.) costly liquids are frequently lost (alcohol and ether in particular) under the form of vapors diluted in enormous masses of air. Often, even in spite of a very considerable recovery, these losses in certain establishments may reach millions of francs per year.

Mr. Claude has proposed for this recovery a method which presents analogies with those which we have previously noted.

I—Air charged with water vapor and with vapors of alcohol and ether is compressed to a pressure such that it is sensibly saturated with water vapor at the temperature which it possesses at the end of compression, and in consequence, such that it will be saturated with water vapor after reaching its ordinary temperature. To realize this result, a pressure of 4 to 6 atmospheres is practically sufficient.

II—The compressed air is cooled in a refrigerator using water. As a result of its super-saturation with regard to the vapor of water, the latter condenses, carrying with it a little alcohol and other vapors, notably vapors of camphor (celluloid industry).

III—The compressed air, relieved of the larger part of its vapor of water, is then carried to the "recoverer." This is a sort of vertical tubular boiler. The compressed air rises from the bottom to the top through the interior of the bundle of tubes. A counter current of cooled air arrives on top of the "recoverer" and circulates from above downward around the bundle of tubes. The compressed air, in rising, meets with air colder and colder. On entering the bundle of tubes, the compressed air condenses out a mixture of water and alcohol. As this mixture can only be solidified at a temperature considerably below zero, there is no danger of an obstruction of the tubes by the solidification of the water. In proportion as the compressed air rises in the bundle of tubes, it deposits mixtures of water richer and richer in alcohol and in ether, that is to say, less and less readily congealed. These mixtures, flowing from above downward in the bundle of tubes, come into contact with regions continuously warmer. The danger of obstruction by ice is then less and less to be feared.

When the compressed air first arrives at the top of the "recoverer," or transformer, it may be subject to a temperature as low as — 90 deg. C., a temperature at which the maximum tension of the vapor of ether is less than 0.5 mm. (0.0197 in.), but which is insufficient to congeal it. The collection of vapors accumulates in the liquid state in the lower receptacle, while the compressed air escapes from the top of the apparatus completely deprived of its vapors.

IV—This compressed air, exhausted of its vapors and cooled, is then carried to the cylinder of an engine where it expands with the production of work. This expansion produces a further marked reduction of temperature in the air. It is this air, at a temperature of — 120 deg. to — 130 deg., which, passed from above downward in the apparatus as a counter current in opposite direction to the compressed air, determines the program of descending temperature of which we have just analyzed the consequences.

V—It is preferable in the apparatus to push the refrigeration to the point where alcohol alone is separated. The ether is then in the way of separation disseminated in the air in the form of small globules. Carried along by the current of air from the bundle of tubes, it is then separated mechanically by its passage through a separator provided with pierced baffles.

VI—The following is a method of operation of the apparatus with regard to the production of low temperatures:

At the beginning all parts of the apparatus being at the same temperature, there is at first in the compressor a warming of air and a disengagement of heat which is absorbed by the circulating water. The compressed air enters the bundle of tubes in the apparatus at ordinary temperature, whence it flows out sensibly at the same temperature. The temperature of this air then becomes lowered in the expansion prime mover. This expanded air returns, chilled, into the shell of the apparatus; there it is warmed before escaping, but it cools the compressed air traversing the tubes. This air then enters the expansion prime mover at a reduced temperature, whence it issues, still colder and with capacity for still further cooling of the compressed air in the bundle of tubes, and so on from one round to another with reducing temperatures.

About one-half hour is required for the realization of the condensation.

With large units, where the initial compression need not exceed 4 kg. per square centimeter (56.89 lb. per square inch), as much as 16 cu. m. (565 cu. ft.) of air may be treated per horsepower-hour.

The Lyons Celluloid Company has made in its Oyonnax factory an installation for treating 200 cu. m. (7063 cu. ft.) of air coming from the laminators and dryers of celluloid. This air, aside from water vapor, is charged with vapors of camphor, alcohol and ether. The apparatus driven by a 25-hp., three-phase motor, giving an initial compression of 5 atmospheres, secures the recovery each twenty-four hours of 150 kg. (330 lb.) of alcohol, 80 kg. (176 lb.) of ether, and 15 kg. (33 lb.) of camphor. In the expansion prime mover the temperatures are: At admission, — 60 deg. C. and at exhaust — 110 deg. C.

There is no danger of explosion. A warm explosive mixture with dry air and ether is not produced until the proportion of ether reaches 50 to 60 grams per cubic meter (0.0021-0.0037 lb. per cubic foot), but in these cases the mixture contains vapor of water and about 20 grams of ether per cubic meter (0.0012 lb. per cubic foot). Furthermore, the mixture is cooled in issuing from the compressor.

II.—Heat Insulators

Fourier's theory of the conduction of heat leads to the following propositions:

1. If a substance has two infinite plane parallel boundary faces, the quantity of heat which, under steady conditions, flows across a surface S from one of these boundary faces to the other is given by the equation:

$$Q = kS\frac{T_1 - T_2}{l}t \qquad (1)$$

where Q = quantity of heat, S = area, l = distance between the two surfaces, T_1, T_2 = temperatures of the two faces, t = time, k = coefficient of internal conduction of heat (thermal conductivity) for the substance.

2. Let us consider two media exterior to the substance under consideration and touching the boundary surfaces of this substance. In these media let us trace two planes parallel to the boundary surfaces of the substance and infinitely near to them. Let us call Θ_1 and Θ_2 the temperatures on these planes in the media which are in contact with the boundary surfaces of the substance under examination. If we consider an area S on these planes at temperatures Θ_1 and Θ_2, we may, as soon as steady conditions are established, write the following relations:

$$Q = a_1S(\Theta_1 - T_1)t = kS\frac{T_1 - T_2}{l}t = a_2S(T_2 - \Theta_2)t \qquad (2)$$

$$\left.\begin{aligned}\Theta_1 - T_1 &= \frac{Q}{St}\times\frac{1}{a_1}\\ T_1 - T_2 &= \frac{Q}{St}\times\frac{l}{k}\\ T_2 - \Theta_2 &= \frac{Q}{St}\times\frac{1}{a_2}\end{aligned}\right\} \qquad (3)$$

Whence adding and solving for Q we find

$$Q = KS(\Theta - \Theta_2)t \qquad (4)$$

in which

$$\frac{1}{K} = \frac{1}{a_1} + \frac{1}{a_2} + \frac{l}{k} \qquad (5)$$

K is the coefficient of transmission of heat from the one of the media at temperature Θ_1 to the other at temperature Θ_2 through the substance of thickness l. It is a coefficient of special importance in the refrigerating industry.

Mr. Biquard of the Conservatoire des Arts et Métiers, at Paris, has studied the various factors on which this coefficient depends.

He gives to the coefficient $\frac{1}{a_1} + \frac{1}{a_2}$ the name *"insulating value" due to the boundary surfaces.* If the conditions on the two surfaces are the same,

$$a_1 = a_2 \text{ and}$$

$$\frac{1}{a_1} + \frac{1}{a_2} = \frac{2}{a}$$

To the coefficient $1/K$ Mr. Biquard gives the name *overall insulating value.*

It is of interest to examine the influence of the boundary surface of the substance. This influence is only negligible when the values of $1/K$ are greater than 4. In this case the "insulating value" of the boundary surfaces is scarcely 5 per cent of the overall value. The latter is in such cases proportional to the thickness of the substance, the coefficient of proportionality k being the reciprocal of the thermal conductivity.

3. The experimental determination of this last coefficient is then of very great importance. Mr. Biquard has made his determinations in realizing, as far as possible, the conditions of Fourier's definition.

To this end he employs the method known in experimental physics under the name of the "guard-ring method."

Let us suppose that in a large plate of insulating substance (flat with parallel faces) we isolate, in thought, a cylinder of section S situated at a great distance from the rims of the plate. Let us assume that, without changing anything in the mode of propagation of heat in the ensemble, we measure the quantity which, under steady conditions, passes from one of these faces to the other of the cylinder of section S. Everything will happen as though the substance studied had infinite parallel faces. The propagation of heat in the cylinder of section S will not be modified by the presence of the sections surrounding it. In this manner, in the experiment, the conditions of the theory will have been realized. The part of the substance which surrounds the cylinder of section S is the guard ring. The breadth of the latter outside the cylinder of section S should be equal to about twice the diameter of this section.

Mr. Biquard has thus realized these conditions:

The insulating substance under examination is placed between two plates of copper maintained, one at 0 deg. C. by melting ice, the other at a temperature constant and definitely determined by means of a current of water. The ice is placed in two chambers separated by an insulating partition; one of these chambers corresponds to the cylinder of section S, the other to the guard ring. The plate of copper which is in contact with the ice, shows, furthermore, a groove limiting the cylinder of section S. The quantity of ice melted in a given time in the chamber which corresponds to the section S is then measured.

We give some of the results found by Mr. Biquard:

1. The smallest coefficient* of internal heat conduction seems to be that of Javanese kapock packed to weigh 15.7 kg. per cubic meter (0.97 lb. per cubic foot).

2. The coefficient of internal heat conduction of a heat insulator increases considerably when it has absorbed humidity. If one takes, for example, granulated cork mixed with tar, its coefficient of internal conduction equals 0.052 when the substance is dry (packed to weigh 275 kg. per cubic meter [17.1 lb. per cubic foot]). This coefficient becomes equal to 0.076 after a period of twenty days soaking in water, as a result of which the cork takes up 310 kg. of water per cubic meter (19.3 lb. per cubic foot) by way of absorption.

*In the advance copy of the paper from which this is reprinted, this value is given as "0.034 meter, calories-ki-log. hour, per deg. Cent." It is not clear what unit is meant. If it means kilogram-calories flowing per hour through a section of 1 sq. meter for a thickness of 1 meter of the insulating material, this and the following figures do not seem to check with published figures at all. We trust that this will be cleared up in time. But as for the purpose of this paper the relative values only of the heat conductivities are of interest, the absolute values are of less importance.—EDITOR.

3. Insulators with intervals of air can only be employed when they can be maintained dry and the air quiet. In such case the most suitable thickness for the layers of air is comprised between 8 and 25 mm. (0.315 to 0.984 in.). If such layers of air are employed, it is indispensable to reduce to a minimum the thickness of solid matter, in directions both parallel and perpendicular to the walls.

From this viewpoint, the "fibrous cement" which, under a thickness of a few millimeters, present a satisfactory mechanical strength, seems likely to give good results.

The Grading Industries—II

BY EDWARD S. WIARD

(*Continued from page* 96)

Screens and Screening

The splitting of loose fragmental or granular material into three or more sizes and according to one or two linear dimensions of an opening, which is commonly called grading according to diameter, is almost universally done by screens; and these devices are by far the most important ones in the whole field of grading. If the components of a fragmental or granular mass are isometric, such as are cubes, spheres, octahedrons, etc., the characteristic of a perfect or theoretical grading resulting from treatment in a battery of screens is the presence of individual particles whose axial* measure is but little less than the width of apertures of the two screens which take part in the grading. If the aperture be square, the axial measures of the limiting grains will be a trifle less than the linear dimensions of the sides of the two squares. If the aperture be oblong, the limiting criterion will be the width of the oblong openings. If the holes in the screens be round, the largest and smallest grains in the perfect grading with isometric solids will have an axial measure but little less than $0.707d$ where d is the diameter of the two sizes of aperture involved in producing the grading. Where the grains are round the limits will be but little less than d.

If the grains be of tetragonal shape, that is, prisms, ellipsoids, etc., with two axes of the same length but the third longer, the criterion of the limits of the grains appearing in a particular grading will be the short axes in the case of square and round holes, or all three of the axes if the apertures in the screen be slotted. Further descriptions of the limits of grading for regular figures might be stated, but these will suffice, for with very little extension of thought they will cover all regular figures.

If a grading be of fragmental material such as would result from crushed rock, then it is evident that if one axis is selected in a particle of such a grading, any other pair at right angles to one another and the first will, within certain limits, vary as to length, the length depending upon the point they intersect the first axis. Also as the first axis is rotated in various planes its length will change within certain limits as well as the two axes at right angles to it. What then is the criterion for the upper and lower limits of grading with material of this kind? It is impossible to state it in concise language; but quite evidently, fixing the mind on one axis as the direction axis in which a fragment is passing through an aperture of the screen controlling the upper limit of a grading, then all the other pairs at right angles to it and for any point of intersection with the first must define all planes which will be bounded by the aperture figure. Otherwise, as is evi-

*The axes are taken in the same position as minerealogical axes.

dent, the grain will be unable to pass through. The axes, of course, do not correspond with the extreme lengths of the planes which can be drawn through them. The extreme upper limit will be a fragment having only one direction axis through which the fragment can pass through the aperture. Evidently somewhere along such axis there is a plane at right angles to it which almost makes contact at three or more points with the sides of the aperture. By a similar course of reasoning the lower limit of a grading of fragmental material can be defined. These definitions are silent as to the shape of the grain. It must be plain on a little reflection that the limiting grains might be roughly cubical or longish prisms or be disks. It is possible to conceive a longish disk of no sensible thickness which would just pass through an aperture in a diagonal position. Later on it will be clear that while such grains may be conceived to report in the undersize of a certain screen they actually, in practical operation, report in coarser sizes.

Shape of Fragments

The eye is the best guide in most cases of commercial screening. Where the grains are more or less isometric the eye will detect variations from the limits of the grading very quickly unless the particles be very small. If the grading contains scaly and longish pieces the eye may be deceived, pronouncing the grading imperfect when it may be just the opposite. With many grading problems in crushed material which tends to fracture in a variety of shapes, good commercial results are often more desirable than theoretical perfection. In the grading of abrasives, for example, what is desired in a grading is cubical or rounded grains of the same general depth of cutting edge, or grains of the same average dimension or roughly of the same volume. On this account, consequently, longish and flattish grains will be more effective if they report with cubical and roundish grains whose diameter is greater than some dimension of the first kind of grains.

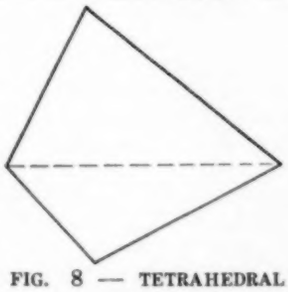


FIG. 8 — TETRAHEDRAL SHAPE

Crushed material like rock or ore yields fragments which are more nearly tetrahedral (see Fig. 8) than of other shapes. The tetrahedral shape is more particularly noticeable where the rock or ore is devoid of cleavage or fracture planes. With irregularly shaped fragments there will, of course, be re-entering angles and many minor sides, but the four planes of the tetrahedron will be found to more satisfactorily bound an irregularly shaped fragment than will a greater number. It is common to assume that rock fragments are more or less cubical but it is rare to find on them six major planes which correspond to the boundaries of a cube. Some writers have considered it convenient to consider all the grains of a grading as cubes having an edge equal to the average of the width of the two screen apertures which control the limits of the grading.

Products of Coarse and Fine Crushing

In the crushing of rocks and ores the useful energy absorbed is proportional to the area of surface made in the crushing. If the surface be calculated as resulting from cubical fragments there will be error, since the individual particles depart widely from this form. In making comparisons between the work of two machines in the point of power consumption, grading analyses followed by computations of the character which have been mentioned will show which machine is consuming the greater proportion of useful energy. It would be expected however that the shape of the fragments would change as the pieces are successively reduced in size. In coarse-crushing operations, cleavage or other pre-existing planes will largely determine the faces in the fragments of broken rock or ore. As the rock or ore becomes more and more finely reduced, pre-existing planes will more and more disappear and the particles will more and more approach ones with perfect isotropism. On this account if comparative tests are to be made with two machines they should be done with the same kind of material.

There is also some variation in the mode in which various crushing machines do their work. Where the machine applies pressure directly to the rock fragments they give way by shearing along planes inclined to the direction of application of pressure. The coarse-crushing machines break in this way, but fine-crushing machines break also by grinding actions to some extent, and the production of surface by grinding takes less power than by shearing. Grinding actions tend to round the grains, and since in the perfectly rounded grain the surface is to the surface of the cubical grain in the ratio of 3.1416 to 6, the cubical rule will report too much useful power absorption. In the fine-crushing machines there are large masses to be moved, either to do the crushing or else power must be employed to move the ore or rock from the point where it enters the machine to the point where it leaves. In some tests made by Phillip Argall at the Independence mill the friction load required for driving a Chilean mill empty was 9 hp.; with a light feed the power increased to 28 hp.; with heavy feeding, to 63; and at the limit of loading the power required was 70. Of course, the bulk of the power used is employed in raising the mullers, and the height to which they are raised increases with the load; but at the same time, as must be evident, considerable increases in power result from moving the greater masses of ore in the mill as the load increases. In coarse or medium-crushing machines the power not usefully employed is almost entirely due to the friction caused by the thrusts of crushing. This is almost exactly the case with rolls running under favorable conditions; for they are perfectly balanced machines.

A much greater source of error in assuming that the average diameter of a grading is the average of the width of the two apertures will be discussed a little later. For theoretical discussions there is no reason why the irregularly shaped particles cannot be assumed by isometric figures of equal volume, and in some discussion which follows the grains are conceived to be spheres, this shape offering the least obstacle to concise statement.

Effect of Overloading Screens

In grading with screening machines and hand screens with material less than ¼ in. in diameter, the lower limit of a grading will appear of very much smaller size than fixed by the smaller of the two apertures. In commercial grading where capacity is desired rather than precision of work, the lower limit of a grading is very much below the size fixed by the aperture. This will be evident by reference to Fig. 9, which shows an extreme case of overloading with a revolving screen and with consequent poor definition of the lower limit of the screening. The sample analyzed by a screening test, and with results as plotted on the figure, is supposed to represent material which has passed through a screen with round holes 18 mm. in diameter and has failed to pass a screen with round apertures 12 mm. in diameter. Only the cross-hatched area pictures grains which

properly belong to the grading, the excess of finer material being due to the inferior work of the screening machine and overloading. In testing the sample with hand screens the individual pieces which remained on the 12-mm. and 8-mm. test screen could readily be picked up by the fingers and tried in different axial positions to see if they were not really particles of undersize. The test work with these two hand screens is consequently 100 per cent or perfect. The sample represents extremely poor work, there being 70 per cent of the material which does not belong in the grading.

In the majority of the rock-crushing plants of the East and the ore-crushing plants of the West, the screening work will average to an efficiency of 60 per cent, there being 40 per cent in different gradings which do not belong with them. In the most precise commercial grading, the best work being done in the abrasive industry, the efficiency of the work does not actually run much over 75 per cent. As shown by hand screens the work is better than this, in some cases over 90 per cent. In making the statement about the efficiency not

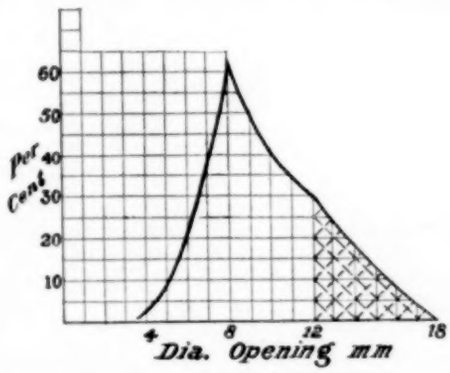


FIG. 9—SIZING TEST ON MATERIAL PASSING SCREEN WITH 18 MM. ROUND HOLES

exceeding 75 per cent I have in mind a very precise mode of measurement, the unaided eye not being deemed a sufficiently accurate guide and hand screens with the finer sizes but little better.

Efficiency of Screening

If the work of a battery of screens be x per cent or any percentage less than 100 or perfect, and some particular grading, such as a $+20-12$-mm. size, be closely probed with screens with apertures differing from one another by 1 mm., there being test screens with 19, 18, 17, 16, 15, 14, 13, 12, 11, 10 mm., etc., holes, what would be the result? There has been no experimental work to tell; but in advance of the theory it can be predicted that the plot for the number of grains resting on each size would be a hyperboia until that screen size was reached. The number of grains would then begin to decline and the curve would take a downward direction from the node and would be a hyperbola with convexity facing that of the first leg of the curve. See Fig. 10, in which diameter of grain is plotted along the axis of X, ranging from 0 diameter at the origin to 20 mm. Axis of Y is plotted for the number of grains resting on the different test screens and of uniform number for sizes above 20 mm. A little reflection will make plain that if the plat is of the form mentioned, any grading of less than 100 per cent perfection will show the greater number of grains in a grading averaging nearer the lower limit of size as defined by the smaller of the two apertures entering into a grading than the upper.

The manner in which the fragmental material has been prepared for the grading will to some extent modify the relations shown by the curves, and they are only strictly true when there would be present in the mass fed to the 20-*mm.* screen the same number of fragments or grains of every size. It is, of course, impossible for the mind to conceive of more than one individual in a grading of fragmentary material of any exact volume. Again the orderly mind will be certain that there will be many sizes lacking. But let it be conceived that all the individuals of a grading can be grouped into three-dimensioned isometric objects of different size, but varying from one another by regular but sensible differences in dimension and that there are equal numbers of such objects; then the plot shown in the figure will follow. The plots are based on percentages of perfect work by weight, from 60 per cent upward. As the screening work approaches 100 per cent the fewer the grains of undersize there will be in a grading and the more nearly will an arithmetical average of the width of the apertures express the average width of the grain in the grading.

Distribution of Sizes in Crushed Material

If a crushing machine be fed with fragments all of which are larger in some dimension than the discharge opening, the screen analysis of the crushed mass will show a heaping up of fragments of sizes near that of the discharge opening, and a very rapid falling off in percentage of material below such sizes. If the material fed to the crusher contains all sizes, but the bulk of it coarser than the discharge opening, then sizes below or near the discharge opening of the machine will not show such a marked falling off in percentage, although there will still be a marked heaping up of percentage in sizes near the discharge opening of the machine.

If after leaving the crushing machine the entire mass is further subjected to comminuting actions reducing it to successively finer sizes, then the heaping up which is found in the coarse-crushing gradually passes to the other end of the plot so that in very fine crushing the bulk of the weight is represented by very fine fragments. See Fig. 11.

The point to be observed for the present is that for any particular grading resulting from the screening of crushed material there may be lacking grains of a certain range of size, and in exploring any particular grading with test screens varying from one another by only slight differences of width of aperture, there might be a heaping up on some of the test screens and reverse conditions on others. It must be plain in any case that

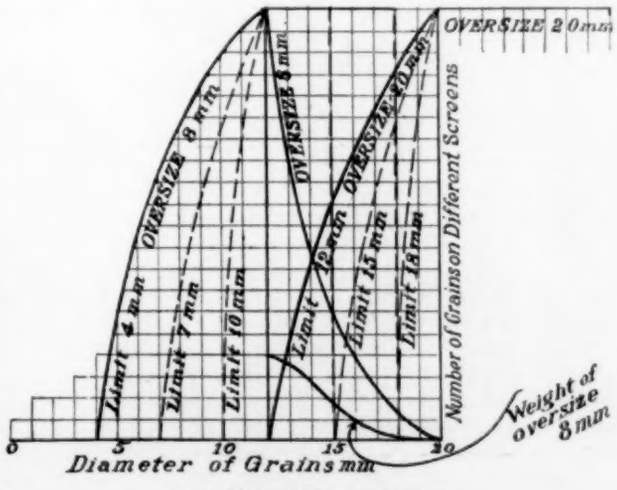


FIG. 10—PLOT OF GRAINS

the arithmetical average of the width of the two apertures does not represent the average size of a grading.

Probability and Chance in Screening

For power-driven screens the question of the degree of freedom with which grains of various size pass through the apertures is one that can best be understood from the mathematics of probability and chance. In the ensuing theory it is assumed that the fragments or grains approach the openings in a layer 1 grain deep and that there is no interference by other grains with a grain approaching an aperture. To put the problem in the simplest form let it be assumed, Fig. 12, that there is a square-meshed opening of length of side l, the thickness of wire or stock between holes being nil. Approaching the border of the square from any direction, and with a rate of speed some elements of which will be indicated later, is a spherical grain of diameter l/n, n being assumed to be of any value equal to or greater than 1. Then evidently for the grain to fall entirely within the square it must pursue some path marked by its center projected, such that at some point in its path the center projected will fall inside the concentric square of side $l - l/n$. A little reflection will show that the chances of the grain falling through the square are in the ratio of the area of the inner square to the area between the inner square and the outer, or

$$\frac{l^2 - \frac{2l^2}{n} + \frac{l^2}{n^2}}{\frac{2l^2}{n} + \frac{l^2}{n^2}}$$

which reduces to $\frac{n^2}{2n-1} - 1$. The inverse ratio fixes the chance of the grain *not* passing through, and is also a measure of the number of squares a grain will have to pass over before falling through.

It will readily be noted that this statement of theory places the problem in its simplest form. It would be expected that if the grain were placed in any position such that its center of gravity were within the square it would fall through the square. All such questions which by extension include impact on the sides of the aperture, the effect of which will depend on many factors, such as the angle of impact, size of particle, velocity of its motion, etc., are not considered, and, as already stated, the question of the interference of the grains with one another is not considered. The question of their interference with one another when screening *en masse* will be touched upon. The mere change of direction by impact does not modify the theory but helps it.

If the square be made with side 1 in., and n be made 8, 4, 2 and 1, the following tabulation can be made of the chance of falling through the square of 1-in. side, and the probable number of squares which grains of ⅛, ¼, ½ and 1 in. will have to cross before falling through the 1-in. square.

Size of Grain	Chance of Falling in Square, Side 1 In.	Probable Squares 1 In. Side to Cross, Proportional to
⅛ in.	3.27	0.31
¼ in.	1.29	0.78
½ in.	0.33	3.00
1 in.	0.00	Infinity

The number of squares to cross is taken under the simplest possible assumption, viz., that the probability of a grain falling through begins *de novo* at each opening and the grains do not follow any regular path.

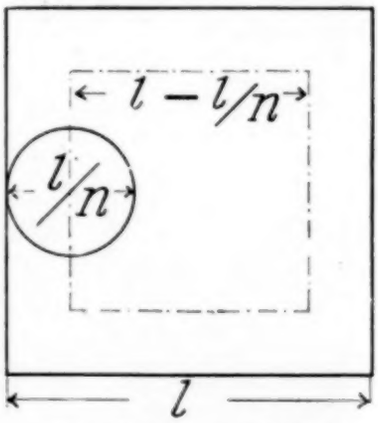


FIG. 12—SQUARE MESHED OPENING

If values of l, l/n be plotted as y and x as the chance, then the hyperbolas which are obtained are symmetrical, the radius of curvature increasing as the value of l increases. The average value of P, or the chance, will represent for any screen having opening side l the average chance of grains passing through from sizes l to 0. Evidently P is the area bounded by the curve between the limits y-equals-infinity and x-equals-l, and the axes of Y and X, and this area divided by l. There can be no quadrature of this area, but an approximate expression can be obtained by considering the curves as equilateral hyperbolas, taking the measure for l in a unit sufficiently small so as to substantially produce a curve of this type, and then considering the curves as tangent to the axes at distances l and to equal the quarter arc of a circle of this diameter. When the average chance is determined from figures of this kind very evidently it increases as l increases. It is in direct proportion to l. A more exact mode of solution would be to take l/n sufficiently small for the first term plotted, so that while the position of x would be at a comparatively great distance from the origin it would be a finite distance. If this be done the value of P can be determined by the methods of the integral calculus, giving as before a value for P which increases as l increases.

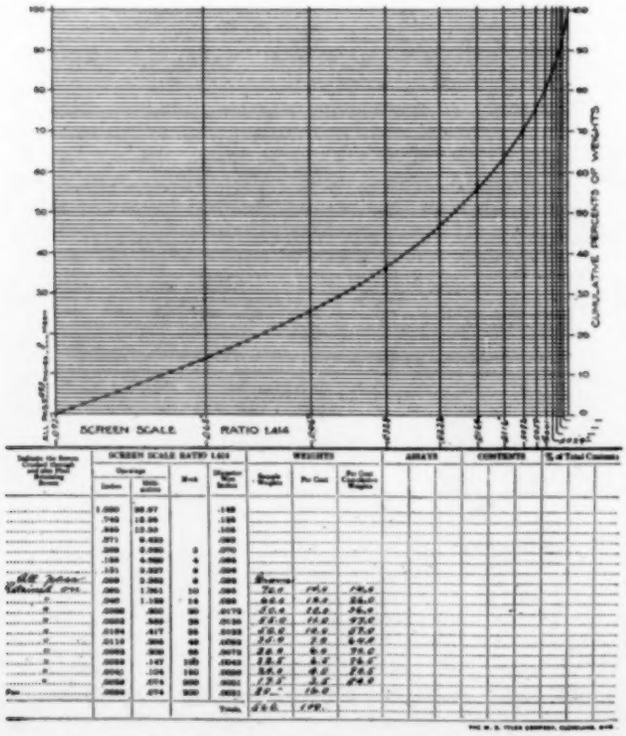


FIG. 11—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS

It has been stated that substances which are comminuted by crushing cannot be reduced in diameter below ¼ micron (a micron is 0.001 mm.), or approximately 0.00001 in. Another way to state the problem would be that, per unit of screen area, the fine screens have less capacity than the coarse, for it will be evident that to get the greater portion of the undersize grains through the fine screens the latter will have to have

greater extent, and this amounts to the same thing as the first statement. Further screening devices of the same pattern and size will have decreasing efficiency or capacity when provided with successively finer screens.

Influence of Other Factors

The question of the velocity of approach towards the screen opening will be considered for the case where the grain approaches the square at right angles to a side and along a line passing through the center of the side. The equation of a fall parabola with origin at the edge of the opening is $x' = \frac{2V^2y}{g}$, where V is velocity of approach of grain in feet per second and g the acceleration of gravity. The maximum permissible velocity with the position of path assumed is obtained when $x = l - l/n$ and $y = l/n$. Substituting these values, V becomes equal to

$$(l - l/2n)\sqrt{\frac{ng}{2l}}$$

When n equals 1 the conditions are obtained when the chance of the grain going through the aperture becomes 0, and V becomes equal to

$$(l - l/2)\sqrt{\frac{g}{2l}}$$

The shortest straight line path will be one which will bring the grain tangent to two sides of the square at some point in the travel, and the longest when the grain pursues a diagonal of the square.

It has already been pointed out to what degree test work with hand screens will permit of tests 100 per cent efficient, or perfect. With fine screens the degree

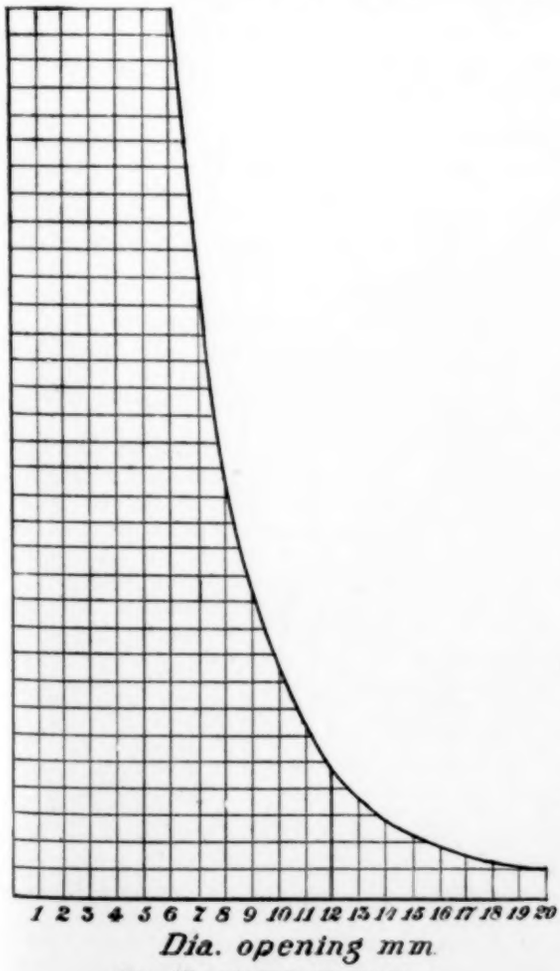


FIG. 13—SPHERE DIAGRAM

of elimination of undersize will depend upon the time spent shaking the individual test screen; but even after prolonged action of this kind there will remain many grains of very nearly the size of the openings. With machine screening the problem is different, for the fragments have only a limited amount of time for treatment, and during this time they must either pass into the undersize or discharge from the screen as oversize. Other things being equal, the screening device which yields the fewest number of undersize grains in the oversizes will be the best. In the curve Fig. 10 I have followed the theory under the assumption that the two screens, the 20 and 12 mm. sizes, will show an efficiency of 60 per cent. Under the theory the 12-mm. screen should show a different efficiency, but the tonnage reaching it is less and these two factors offset one another. No attempt has been made to measure the effects of these two opposing factors.

In Fig. 13 a curve is shown of the number of spheres below 20-mm. size required to make one sphere of 20-mm. size. From these data the weight curves on Fig. 10 have been computed and plotted.

Interpreting the Chance Law

Some further explanation may clear up some difficulties which may arise in the reader's mind. It has been stated that one ratio of the areas which have been mentioned represents the chance of the grain going through an aperture, while the inverse ratio represents the chance of its failing to go through an aperture. It has been stated further that this ratio is also a measure of the number of apertures which the grain has to cross before being eliminated if it is an undersize grain. Now on the plot for the 20-mm. screen the lower limit of undersize grains which will pass into the oversize has been fixed at 12 mm. If there be barely enough apertures passed over to eliminate all the 12-mm. grains, then all grains larger than this should report in the oversize of the 20-mm. screen, for the larger grains have not had enough apertures to pass over. I believe the reader will overcome this difficulty if he considers that while the inverse ratio is a measure of the number of squares to cross, it is the probable number of squares to cross for *one* grain.

The proper conception of the plot will be obtained from the following considerations. Let it be assumed that any particular size represents the lower limit for the oversize, and that there are A grains of each size, including the lower limiting size and the ones larger. Let x be the area of the chance inner square for any particular sized grain, and y the area between the inner and bounding square of the aperture. Then of A grains approaching a line of apertures of the lower limit of size, the fraction $x/x+y$ will pass through the screen and the fraction $y/x+y$ will remain in the oversize. These will be the fractions removed and remaining respectively after passing the first line of holes. Of the grains remaining there will be eliminated at the second line of holes a similar fractional part $x/x+y$, and so on, until after a sufficient number of lines of holes have been passed, all the lower limit of size has been removed. For the same number of lines of holes the proportion of undersize grains of larger sizes can then be determined, and on this being done a curve will be obtained similar to those of the oversize portions of Fig. 10. The undersize curves can be obtained by difference. I believe this explanation will also cover the difficulty in interpreting the chance law from the curves. As the law has been stated at an earlier point, there should always be fragments of all sizes in the oversizes and the plot of an oversize should pass through the origin; but the chance law is only for one aperture, and

in actual screening the grains pass over many apertures, each removing its burden of grains which do not belong in the oversize; consequently the chance law for actual screening operations must be taken in the manner explained above.

The limit of the undersize grain which does not belong with an oversize grading may be raised to any desired amount. For the 20-mm. screen, the lower limit instead of being placed at 12 mm. can be placed at a higher point between this one and 20 mm. Curves where the limit is at 15 and 18 mm. are shown. For the 12-mm. screen the limit instead of being at 4 mm. can be placed anywhere between 4 and 12 mm. Curves are indicated for 7 and 10 mm. limits. The position of the curves depends upon the number of apertures passed over. Theoretically only when an infinite number of squares have been passed over can the work of the screens be perfect, when the curves of oversizes would be straight lines. If all the grains are not treated alike, that is, do not have the same opportunities for passing over apertures, then the plot will not follow. This discordance will arise when a screen is heavily overloaded. In this case exceedingly fine grains often do not have a chance to come into contact with the screening surface at all and pass into the oversize without having undergone any screening action. Note the minus 200-mesh grains of Fig. 11, illustrated on a Tyler cumulative diagram of a screen analysis.

Unit Capacity for Flat Screens

In order to determine the capacity of flat screens for varying sized feeds, uniform conditions must be imagined. If the adjustments of a screen to which material averaging ½ in. diameter is fed are (1) length of stroke 1 in., or somewhere proportional to the diameter of the largest material fed, and (2) number of strokes 200 per minute; then if 1/50-in. material be fed it is quite evident that to give the grains equal possibilities as to chance the stroke must be reduced to 1/50 in. If this be done, the only way the capacity of the screen can be made equal to the first case would be to increase the number of strokes 50^2 times 200—an impossible number and contrary to the opening statement of this section. If the length of stroke be reduced to 1/50 in. while the number is maintained at 200, then evidently the capacity is only $(1/50)^2$, or, in other words, the capacity is proportional to the diameter squared. If the length of stroke be maintained at 1 in. and the number of strokes be as before, 200, then evidently the capacity of the screen is proportional to the diameter. The efficiency of the screen will, however, be markedly inferior to that where it is being fed coarse material. If the diminishing average chance be considered, which falls off from size to size directly as the diameter is reduced, then evidently when all factors are equal the capacity of a screening machine varies as the cube of the diameter of particle fed. Commercial figures of capacity lie between those proportional to d and d^3. With regard to the fine sizes it is to be observed that the eye will not detect much error in the oversizes, and a hand screen is not much superior in this respect. In the abrasive industries all that the trade demands is that the grains of a grading appear of about the same size.

Ordinary shaking screens have an advancing speed not to exceed 50 ft. per minute. The number of strokes does not exceed 300; consequently to attain an advancing speed of 50 ft. per minute the stroke has to be 2 in. or better. In actual practice it is nearer to 25 ft. per minute.

Capacity of Belt Screens

The Callow belt screen has belt speeds ranging from 34 to 100 ft. per minute. The capacity for a duplex screen made of two endless screen-cloth belts, each having a width of 2 ft., is given by the makers as follows:

Mesh	Opening, In.	Capacity, Tons 24 Hrs.
20	.027	250
30	.0145	200
40	.014	150
60	.00916	125
80	.00675	100
100	.0055	75
150	.0041	50

These figures apparently have been determined by taking the capacity of the screen belt at the maximum speed, the grains being considered to be one deep. It follows roughly the rule for diameter. In determining the capacity of flat shaking screens, I believe 10 tons per day of twenty-four hours per square foot of screen surface, for a product ranging in size from 1 in. to 0, represents the average load. For the most precise work this figure can be regarded as the base figure also, and the square root rule followed. Under the direct ratio rule a 12-mesh screen with 0.054-in. opening would have a capacity per square foot of 0.027/0.5 times 10, or 0.54 ton per square foot of surface per twenty-four hours. Under the square rule the capacity would be $(0.027)^2/(0.5)^2$ times 10, equals 0.029 tons per square foot of surface per twenty-four hours. For a 4-ft. by 5-ft. screen the capacity with inch material would be 200 tons per twenty-four hours; for 12-mesh material, by the direct rule, the capacity would be 10.4 tons, and by the square rule, 0.6 tons per twenty-four hours. These rules are only intended to apply to problems where there are successive screening operations from coarse to fine. If the problem be the use of a single screen of fine mesh to remove fine particles from a mass containing all sizes from coarse to fine, the average diameter of the particles fed will govern the capacity.

Some factors affecting capacity I do not deem worth while touching upon. Friction would be greater with the fine screens, if the apertures of a screen be considered pipes for the transport of material, for even if the stock of which the screen is composed is reduced in thickness to a degree proportional to the width of the aperture, since the number of apertures per unit of surface increases as the reciprocal of the square of the opening, the opposition to discharge must increase as the linear number of meshes increases. As a matter of fact the ratio of the weight of stock in the screen per unit of area to the area of opening increases as the number of apertures per linear unit increases, and consequently the increase of friction as the size of the apertures is diminished is something more than can be expressed by a simple ratio.

With most fragmental material produced by crushing there is an excess of the finer sizes, so with screening machinery uniform as to stroke, shape, etc., what may be proper in the way of a screen at the head of a single line of screens will show overloading for finer sizes under the rules given. But such discrepancies will have to be disregarded in practical designing, the capacity being fixed for the screen with largest apertures and the rest doing the best they can. In this connection it will be stated again that while with coarse sizes the eye will detect variations in the diameter of a very small percentage among the different pieces of an oversize, with the fine sizes it will be ignorant of variations amounting to 100 per cent or even greater. In testing with hand screens the individual screens are shaken until the eye is satisfied; and the eye being the final criterion of commercial screening, much greater capacity is satisfactorily obtained than would be permissible if exact criteria of the efficiency were used.

If a screen of any particular size receives a feed which covers it one grain deep, then if the width be diminished by a half the screen must be covered with a layer

two grains deep. If now the screen be made twice the original length then evidently the second arrangement of surface is as efficient as the first, other things being equal, unless the proportion of oversize in the feed to the screen is very great. Where the proportion of oversize of a feed is large, a broad short screen is called for. For usual screening problems a ratio of length to width of 2 : 1 is preferable. The principal objection to wide flat screens is the difficulty of distributing the feed over them. If proper distributing means are adopted there is much loss of head room. One of the principal advantages of horizontal or nearly horizontal revolving screens is that they require no distributing devices, the material to be fed being merely dumped or fed in at any convenient point. Capacities of revolving and other types of screens will be discussed later.

Denver, Col.

Lutes and Cements

A paper read at the Baltimore meeting of the American Institute of Chemical Engineers.

BY S. S. SADTLER

I use the words *lutes* and *cements* together for two reasons. First, because they have different meanings, the former referring to temporary applications and the latter to more permanent uses. Second, the use of these words conjointly shows to a casual reader that the subject of this paper is not solely with reference to Portland cement.

Quite a few years ago I read several papers on the present subject. In these papers I classified the formulæ according to composition in one instance and according to intended use in another. These papers were noticed by a number of people who communicated with me, particularly engineers.

It is very difficult to give proper credit for what one finds in the literature, as some of these formulæ are like humorous stories that are rarely traceable to a source. In looking up this subject for the present occasion I noticed some eight or ten attractive formulæ in a well-known handbook of industrial chemistry. These were credited to a periodical which I soon found quoted one of my papers, and I was not the originator of more than a few of them. Thus, things are passed on. The value of the formulæ was diminished, however, by reason of the space allotted and the changes made.

I will classify the formulæ now as I did before the Engineers' Club of Philadelphia, by the use of the following subdivisions. Very few of the formulæ given herewith have not been tried by me. The only instances are where the authority was excellent and adequate trial was not practicable for some reason.

1. Waterproof

Of the chief bituminous substances available we have pitches, asphalts, gilsonite and blown petroleum residues, mixtures of these substances and mixtures of the same with inert material.

(*a*) Of the asphalts I prefer a substance like refined Bermudez, as it is nearly pure bitumen and forms homogeneous solutions with suitable solvents. A minor percentage of boiled linseed oil or blown petroleum oil is useful sometimes for tempering or fluxing. For solvents heavy naphtha is generally used. It does not dissolve all the asphalt, such as the so-called asphaltene portion, but it thins out the asphalt well enough. Coal-tar naphtha is better but more costly. As voids are very apt to form in a painted asphalt coating, some form of filler is generally employed, either by coating paper so that it becomes more or less impregnated or by the addition of a filler, such as silex, infusorial earth, etc. The natural mineral filler of Trinidad asphalt may serve the purpose of such filler.

(*b*) For a soft, water-tight coating in cement work, blown asphalt and infusorial earth, thinned with a solvent, such as heavy naphtha, is employed.

(*c*) The use of blown asphalts with natural asphalts and gilsonite is in some cases desirable, as these products are ideal fluxes for hard asphalts and tend to prevent brittleness or separation of the films when the solvent evaporates. The harder or higher melting point grades of these blown residuums are capable of making good cements alone or with filler, but are rather hard to dissolve. Such artificial asphalts have melting points as high as 150 deg. C., while the softer ones that are probably best for fluxing have melting points about 80 deg. C.

(*d*) Lutes of boiled linseed oil, thickened with clay, asbestos, red or white lead, etc., are waterproof, if thick enough from the filler added.

(*e*) Flaxseed meal made into a stiff paste with water is useful as a lute for steam connections and is easily applied.

Portland cement only serves as a waterproof cement when given time for the preliminary setting to take place. It is not generally impervious to water, and because of its colloidal character while setting it seems hard to realize that it could ever act other than as a water pervious diaphragm. But when fully or reasonably well set and dry the colloid character no longer exists and does not return to these particles that have taken part in the setting from hydration.

For all practical purposes the use of trade preparations, such as those containing metallic soaps or oil emulsions, serves to render concrete approximately impervious, but I will refer to a few points I have noticed in making or using lutes and cements.

(*f*) For making small experimental electrolytic cells, etc., for demonstration purposes, I have found it desirable to use white iron-free Portland cement and white plasterers' sand that had been sieved. If a one-to-two mixture is made with the prescribed quantity of an approved waterproofing substance, such as the product of the Newberry patent, and care is taken to work out voids and the article in the *green* is kept moist, a very nice waterproof container can be made. It will also stand dilute acids or alkalies, but not the two alternately. For large articles wire-mesh reinforcement is often desirable, and if the vessel is used for dilute acids this might well be made of a mesh of Monel metal, etc.

(*g*) In the literature on this subject, if it can be dignified by that appellation, one finds frequent reference to making lutes with Portland cement and silicate of soda. Some of these may work and others will not, and I approach them with caution. A little silicate of soda in the water will help toward a quick initial set, for instance, for under-water work. Water containing 4 to 20 per cent of silicate of soda by volume causes a preliminary set in 2 hr. that prevents washing away by water. It injuriously affects the natural and gradual hydration, however, unless the sodium hydroxide, carbonate or sulphate is washed out by excess of water, which is difficult to do. Mixtures containing above 4 per cent of silicate show strong efflorescence, which is particularly marked with that made with 20 per cent aqueous solution of silicate. The silicate-of-soda-setting is at first due purely to double decomposition of sodium silicate and calcium hydroxide. If very strong silicate is used the mass stiffens so fast that it cannot be worked. This is true of all concentrations of 50 per cent or over.

There is one point that might be noted. For immediate immersion in running water silicate of soda acts

as a binder, while the particles of the cement form gelatinous colloids instead of being washed away. The soluble sodium salts are more or less eliminated and apparently normal concrete results. I have used glue solutions (3 to 5 per cent strength) with white Portland cement to secure very dense compositions. It is possible that the glue aids colloid formation, and protects the same against too rapid crystallization.

2. Oil Proof

(*a*) Several of the best known lutes I am placing under this category, although they might be classified elsewhere:

Good glue	2	parts by weight
Glycerine	1	" " "
Water	7	" " "

The glue is first softened by the water, then liquified by heat and the glycerine incorporated. This is a good lute to render corks vacuum tight, for stopping small leaks of almost anything except water and steam.

(*b*) The following has been found useful in laboratories and in the works for oil vapors:

Putty of molasses and flour.

(*c*) The next lute to be mentioned is one of the best known and most useful. The proportions given are those I have found satisfactory:

Glycerine	90	parts by volume
Water	10	" " "

To be made into a stiff putty with

Litharge	90	parts by weight
Red lead	10	" " "

It takes several hours to stiffen and about a day to set.

(*d*) Several cements of which silicate of soda is an active principle or the chief binding substance are as follows:

Silicate of soda (about 30 deg. Be.)
Whiting

Made into a stiff putty.

This slowly sets by drying if not by chemical action. If magnesium carbonate be used the setting is so quick that it is hard to use the mixture. If it is used, however, the silicate should be diluted to one-third to one-half normal strength. I do not recommend either of them.

(*e*) Barium sulphate is often used with silicate of soda. The silicate should be about 30 deg. Be. If the mixture is heated by a preliminary heating of the silicate or it is put in a hot place there seems to be some chemical reaction which aids in the setting.

(*f*) A very strong and ultimately hard cement or substance for a variety of purposes is made by incorporating glue with plaster of Paris. This will be described with other plaster plastics in Section 6.

3. Acid Proof

(*a*) The asphaltic preparations referred to in Section 1 are largely acid proof. Black putty, the formula for which was first noticed by me in a "Handbook of Chemical Engineering," by George E. Davis, is made by intimately mixing equal weights of China clay, linseed oil and gas-tar. The ingredients must be anhydrous, so it would probably be best to take a tar residuum or very soft pitch of thick, so-called, creosote oil. Rubber cements may have varied composition, but I will only refer to two which are, perhaps, extremes. If equal parts of fresh unvulcanized rubber are used the mass is so stiff that it would be probably used alone. If as much as four parts of linseed are used considerable filler can be incorporated and make a workable putty.

(*b*) Equal weights of rubber and boiled oil are taken; the rubber is first dissolved in carbon disulphide in the proportion of 4 c.c. carbon disulphide to 1 gr. of cut-up rubber. Boiled linseed oil is then mixed in, and the mixing is facilitated if the oil is warm. The solvent is generally not removed by evaporation until the paste is applied.

(*c*) The other formula, to which reference has just been made, differs in having four times as much boiled linseed oil and then fire clay or other filler, such as silex, is used.

Crude, finely cut rubber	1	part by weight
Linseed oil, boiled	4	" " "
Fire-clay	6	" " "

(*d*) Melted sulphur with fillers of stone powder cement, sand, etc., are used. The following is used for hydrochloric acid vapors:

Rosin	1	part by weight
Sulphur	1	" " "
Fire-clay	2	" " "

Linseed oil (boiled) and fire-clay stand most acid vapors.

(*e*) According to Davis red lead and litharge in boiled oil stand acid vapors (even nitric acid).

Litharge	80 lbs.
Red lead	8 "
Flock asbestos	10 "

Fed into a mixer, a little at a time, with 6 qt. of boiled linseed oil.

There are numerous acid-resisting mixtures possible in which slicate of soda is used. This is possible, in spite of the strong basic character of silicate of soda, because the silicate is superficially changed to colloidal silica, which continues the cementing work, at first effected by the silicate of soda.

(*f*) Barium sulphate, powdered glass, China clay, etc., are used with silicate of soda slightly diluted with water. A strength of about 30 deg. Be. is close to what is best for the purpose.

(*g*) The following I have used with success for dilute hydrochloric acid:

White China clay	1	part by volume
Fine white sand, or powdered quartz and sand	2	" " "

are mixed thoroughly and worked up with just enough silicate of soda, diluted with an equal volume of water to make a paste. This can be rendered more impervious to water by the judicious incorporation of organic colloids. If a little fine casein be incorporated with the silicate of soda in a mixer so that the mixture is quite smooth the mass will be better. About 5 per cent of fine, dry casein powder is added, based on the weight of silicate. If fresh milk curd can be used, corresponding to the same dry weight of casein and allowance is made for the water contained, a mechanical mixer may not be needed.

4. Chlorine Resistant

(*a*) The most reliable are made with Portland cement as chief ingredient:

Powdered glass	1	part
Portland cement	1	"
Silicate of soda	1	"

The last mentioned should be diluted considerably so as not to set too fast.

Portland cement quickly reacts with silicate of soda, but powdered glass and clay react more slowly with silicate of soda, but finally render the cement insoluble as regards its mass.

5. General Purposes with Plaster of Paris

This series of lutes and cements is highly important, and individual formulæ will serve to prevent the escape of hydrocarbon and other gases in furnace work, as cements for mechanical purposes and as coatings, such

as plaster, when mixed with asbestos straw, plush trimmings, hair, etc.

(*a*) Soluble sulphates form double sulphates with calcium sulphate and with water. They set harder and are more impervious than calcium sulphate (plaster of Paris) alone. It is desirable not to take equal molecular quantities to form the full double sulphates, but about half of these quantities. Sodium, potassium and aluminum sulphates are used for this purpose.

(*b*) According to Sigmund Lehner, a little borax in the water used makes hard cements and regulates the setting: 12 volumes water to 1 volume saturated borax solution sets in 15-20 min. at 10 deg. C.; 8 volumes water to 1 volume saturated borax solution sets in 1 hr. The same author gives a formula credited to Viotti for a weatherproof plaster cement.

(*c*) One thousand five hundred grams of borax and 150 grams of magnesium oxide are melted together and powdered. This powder is then mixed with 75 grams of plaster of Paris. Borate of magnesium thus predominates and protects the plaster from being washed away by water.

6. Marine Glue

Standard preparation of this class of lutes, (which are applied to crevices, etc., hot, and get firm but not brittle when cold) is composed as follows:

Crude rubber	1	part by weight
Shellac	2	" " "
Pitch	3	" " "

The rubber is first dissolved in carbon disulphide or turpentine before mixing with the heated (not superheated) mixture of the other two. The advent of blown petroleum residuums has made it possible to make up hard but flexible compounds without rubber. Grahamite is a good base to which fluxes, such as these just mentioned or soft asphalts, are added.

7. Gasket Compositions

There are so many good gasket compositions on the market that I was on the point of omitting all reference to them. For factory work I have found no trouble in getting ropes of graphitized asbestos, rubber composition leather fibers cemented with rubber, etc., that are quite satisfactory.

In the laboratory one can generally make out for low temperatures and pressures by saturating heavy "kraft" wrapping paper with soft pitch, such as wood pitch for steam or with gelatine and glue (hectograph) composition for oils. For high pressures, slots filled with lead rings and a *V*-shaped rim to the lid are most satisfactory.

8. Machinists' Cements

A few words might be said here with reference to machinists' cement. These are the well-known red and white leads. The red lead is often diluted with an equal bulk of silica or other inert substance so as to make it less powdery on drying. The best way that I have found to accomplish this, however, is to add rubber or gutta-percha to the oil as follows:

Linseed oil	6	parts by weight
Rubber or gutta-percha	1	" " "

The rubber or gutta-percha is dissolved in sufficient carbon disulphide to give it the consistency of molasses, mixed with the oil, and left exposed to the air for about 24 hr. The red lead is then mixed to a putty. Oxide of iron makes less brittle cements than red lead.

9. Leather Cements

As cements of this class are apt to be wanted by chemists sometimes, I quote from my paper in the *Transactions* of the Engineers' Club of Philadelphia, *XXI*, 4, 1904.

The following formulæ are given in the "Papier Zeitung," Vol. 18, p. 2618:

"(*a*) Equal parts of good hide-glue and American isinglass, softened in water for 10 hr. and then boiled with pure tannin until the whole mass is sticky. The surface of the joint should be roughened and the cement applied hot.

"(*b*) One kilo of finely shredded gutta-percha digested over a water-bath with 10 kilos of benzol, until dissolved, and 12 kilos of linseed oil varnish stirred in.

"(*c*) Seven and one-half kilos of finely shredded india-rubber are completely dissolved in 10 kilos of carbon disulphide by treating while hot, 1 kilo of shellac and 1 kilo of turpentine are added, and the hot solution heated until the two latter ingredients are also dissolved." Precautions against fire and vapors should be observed.

(*d*) Another one noticed in the "Journal of the Society of Chemical Industry":

Gutta-percha	8 ozs.
Pitch	1 "
Shellac	1 "
Olive oil	1 "

These are melted together.

10. Iron Cements

When iron in a fine state of division, as in fresh oil-free filings or cast-iron borings that have been powdered, is mixed with an oxidizing agent, such as manganese dioxide or a substance electro-negative to iron, such as sulphur, in a good conducting solution like salt or salammoniac, galvanic action sets in very rapidly, ammonia is given off (if salammoniac be used) and the iron swells, by forming iron oxide, and cements the mass together. It is best diluted with Portland cement. A formula which I give from memory is:

Iron filings	40 parts
Manganese dioxide, or flowers of sulphur	10 "
Salammoniac	1 "
Portland cement	20 to 40 "

Water to form a paste.

These cements are used extensively in foundries, etc. I prefer manganese dioxide to sulphur as a negative element. The free carbon of cast iron that has been powdered is negative pole enough to react, but the use of manganese dioxide causes quicker action. If not diluted with cement they are apt to expand too much and bulge out from the hole or crevice in the iron.

11. Crucible Cements

(*a*) Mixtures of clay and borax in which the clay predominates. Silicate of soda and powdered glass or sand are the best known compounds for cementing lids on crucibles, etc. Sometimes the "iron cements" are used for such purposes, or iron filings and a little manganese dioxide are added to the above compositions for crucible cements.

(*b*) Probably the best known cement for graphite is fire-clay, which with water binds the graphite fairly well and itself stands high temperatures.

(*c*) In some cases it is desirable to have an all-carbon binder, and for this purpose tars or soft pitches are used. Very little binder must be used, however, or the material will crack when heated, as the carbonizing of the pitch shrinks the binder somewhat. Starch paste has been recommended for this purpose, but the shrinkage is greater and the binding is not as good.

(*d*) A strong, waterproof cement that will stand high temperatures may be made by mixing powdered silica or fine sand and powdered silica with a solution of magnesium chloride of about 10 per cent strength. This composition is applied as a putty and then painted

or soaked in a solution of silicate of soda of about 30 per cent strength. This forms magnesium silicate as a binding material for the silica.

13. Magnesia Composition for Furnaces, Etc.

Magnesia burned at incandescent heat or hard-burned, 80 per cent. Magnesia light burned (just sufficient to drive off carbon dioxide (dull redness), 20 per cent. The composition is made into a stiff putty with water and shaped as desired. The water must be driven off very slowly. A small proportion of good asbestos fiber may be worked in to keep from cracking.

14. Oxy-Chloride Cements

These are sometimes called stone cements and the only one of practical importance is that made with magnesium chloride.

If the magnesium chloride is quite pure, i. e. free from potassium or calcium chloride, the solution used need not be more than 18 deg. Be., but if commercial German magnesium chloride be used it requires a solution of 20 to 22 deg. Be. The magnesium oxide must be freshly burned at a dull red heat as the light oxide that is requisite for this cement readily absorbs carbon dioxide from the air and becomes unsatisfactory or of no use for making cement compositions. A diluent for the magnesium oxide is generally used, such as silica or wood pulp, ground wood, etc.

This communication is not offered as being an exhaustive treatment of the subject, but as an attempt at bringing together only tried or traceable formula for the making and using of lutes and cements.

Science and Aesthetics

From Dr. John A. Brashear's presidential address before the American Society of Mechanical Engineers (Dec. 7, 1915) we cull the following paragraphs:

"Dr. Maclaurin tells us a story of Matthew Arnold, the apostle of sweetness and light, in which he is made to say that when a candle burns the oxygen of the air combines with the carbon in the candle to form carbonic gas, *but who cares?* The story is told because it suggests an attitude to science that is far from rare, even among people of intelligence to-day. He recalls the poetic query of Keats when he writes of the rainbow: 'Do not all charms fly at the mere touch of philosophy? There was an awful rainbow once in heaven, we know her woof and texture. She is given in the dull catalog of common things. The complaint seems to be that science with its analysis of color robs us of the pleasing sense of awe and mystery, but if you dig deep you will find enough mystery left to satisfy the keenest yearner after half lights and the obscure. At best, science replaces one mystery only by another of grander order.'

"Now, leaning for a moment from the purely utilitarian side of science to the aesthetic, there comes a joy to the lover of the beautiful that cannot be expressed in words, for he sees in the color of an American Beauty rose the same light waves that tint yonder red star, whose light waves, coming to him at the rate of 180,000 miles per second, left it a thousand years ago. He listens to the varying bell tone of the swiftly moving locomotive and translate it into the motion of stellar worlds, whose distances have never before been measured by the most refined modern instruments.

"Does the photographic picture of your favorite landscape lose any of its beauty when you are told that during an exposure in your camera of only one-tenth of a second from forty to eighty millions of millions of light waves have hammered upon your negative tending to shatter them to pieces or change their molecular arrangement?

"Maclaurin gives a charming illustration to help us better understand what it means when a ray of violet light impinges on our photographic plate for one-tenth of a second. 'Imagine you are watching a log floating near the seashore, and that it strikes against a pier as it rises and falls with the waves, say, once in six seconds. In order to correspond to the number of light waves in one-tenth of a second, the log would have to beat up against the pier for more than two million years.'

"But enough for the present of the aesthetic side of science, though it has beauties in its every phase, whether it be in the flower or in the rainbow—whether it be in the structure of glass from which the prism is made, or the story it tells of yonder far-off universe. But can we connect this science beautiful with our engineering problems? Why not?"

Analysis and Assay of Zinc Retort-Residue

The smelting of Western zinc ores leaves a retort residue which is sometimes valuable for its content of gold, silver, lead or copper, or some combination of these metals. It is customary to resmelt these residues in lead blast furnaces. In some cases the residues are handled by the company originally selling the zinc ore, and in others by the zinc-smelting company itself. In either event, the practice has brought new problems to the analytical chemist in determining the valuable constituents, on account of the large quantity of carbon left in the residue and the refractory character of the sintered material. In the early history of residue analysis, chemists found difficulty in obtaining concordant results and in checking each other, due largely to the fact that the methods applicable to ores were not wholly suited to retort residue, as well as to a lack of uniformity in the methods adopted. These difficulties have now been largely overcome, and we present some of the methods adopted at various places. Constructive criticism is invited in the hope of making the methods more exact and efficient.

Details of Analysis

The following methods are in use at the Hillsboro, Ill., plant of the American Zinc Company of Illinois:

Carbon.—Weigh 1 g. of the residue into a casserole, add 15 c.c. HCl and boil five minutes. Add 30 c.c. hot water, clean the casserole thoroughly and filter through a weighed filter, washing with hot water. Remove the filter and contents and dry at 100 deg. C. Weigh and ignite until all carbon is burned. Weigh the ash, and calculate carbon as the difference between the weight of the ash plus filter paper and the weight of dry carbon plus filter.

Zinc.—To the filtrate from the carbon determination, add 15 c.c. ammonia and 1 c.c. hydrogen peroxide, and boil until all peroxide is expelled. Filter and wash with hot water. Dissolve the precipitate in the original beaker with a little HCl and reprecipitate as before, filtering through the same filter. To the filtrate add 20 c.c. HCl, and if copper is present remove it by adding 10 g. test lead and boiling for ten minutes. Determine the zinc volumetrically by titration with standard solution of potassium ferrocyanide, using ammonium molybdate as indicator.

Iron.—Dissolve the precipitate, obtained by the above treatment, in the original beaker with dilute HCl (1 : 3). Wash and boil. When boiling hot add, drop by drop, a saturated solution of stannous chloride until the solution is clear. Cool and add 25 c.c. saturated

solution of mercuric chloride. Determine iron volumetrically by titration with a standard solution of potassium bichromate, using a weak solution of freshly prepared potassium ferrocyanide as indicator.

Sulphur.—Weigh 1 g. of the residue into a 50 c.c. crucible and mix thoroughly with 10 g. Eschka's mixture (two parts MgO, one part Na_2CO_3). Heat slowly over a Bunsen burner until all carbon is expelled, stirring frequently with a platium rod. Dissolve the mixture in dilute HCl, add 1 c.c. hydrogen peroxide, boil and filter, washing the residue thoroughly. Precipitate sulphur as barium sulphate by the addition of barium chloride, and boil for five minutes. Allow to settle and then filter, wash, ignite and weigh as barium sulphate. The weight multiplied by 0.1375 gives the equivalent of sulphur.

Lead.—Weigh 2 g. of the residue into a 200 c.c. flask and add 10 c.c. HNO_3, 6 c.c. HCl and 10 c.c. H_2SO_4. Boil to white fumes, cool, dilute with 50 c.c. water, boil, filter and wash. Dissolve the lead sulphate thus obtained in a clean beaker with hot ammonium acetate, and after washing thoroughly add 20 c.c. of a 10 per cent solution of potassium bichromate. Allow to settle and then filter, washing thoroughly. Place the filter and lead chromate in a clean beaker, add 50 c.c. water and exactly 1 g. ferrous ammonium sulphate. Add 50 c.c. HCl and titrate the excess iron with standard solution of potassium bichromate. Lead equals 1.226 times the difference between total iron in 1 g. ferrous ammonium sulphate and the amount found by titration.

Copper.—To the filtrate from the above add an aluminium cylinder large enough so that one-third of its surface will not be immersed in the solution. Boil until all copper is precipitated; remove the cylinder, rinse thoroughly and dissolve the adhering copper into a clean beaker. If any copper remains behind, filter and add to that dissolved from the cylinder. Dilute the solution, add a slight excess of ammonia and titrate with standard solution of potassium cyanide.

Determination of True Silica

The exact determination of silica (SiO_2, not insoluble) requires unusual precautions under ordinary circumstances, but seems to offer particular difficulty in retort residue. The method used at Hillsboro is as follows:

To 1 g. of the residue add 20 c.c. aqua regia and run to dryness. Bake in order to dehydrate soluble silica. Cool, add 15 c.c. HCl and boil five minutes. Dilute with 30 c.c. hot water, filter and wash. Place the filter and contents in a platinum crucible and burn off the carbon. Mix the ash with 10 g. Na_2CO_3 and fuse. When the fusion is quiet, cool and remove to an evaporating dish, preferably platinum, and dissolve the melt with water and HCl. Evaporate to dryness and bake until no odor of HCl can be detected. Add hot water and HCl and filter through an ashless filter, washing until the filtrate shows no chlorine. Ignite and weigh as SiO_2.

At the Ohio & Colorado smelter, where retort residue is smelted, the procedure for silica is slightly different. One-half gram of the sample is mixed with 6 g. of sodium peroxide in a 20 c.c. nickel crucible, and covered with 2 g. sodium hydroxide (a stick about ¾ in. long). Fuse in an assay muffle at a faint red heat. The action once started is violent and soon complete. Allow the crucible to stand for three minutes at a temperature slightly above the melting point of the mixture, finally removing it to a casserole to cool. Cover with a watch glass, and cautiously fill the crucible with cold water. When the action ceases, fill the crucible again, after which the melt should be completely dissolved. Remove and carefully wash the crucible. Now add HCl until the solution clears. If black particles are visible in a residue, the decomposition has been incomplete and the assay must be repeated. Evaporate to dryness, but avoid baking so hot that a black color appears. If the residue is a light brown, the operation has been properly conducted. Cool, moisten with 20-25 c.c. conc. HCl, add 100 c.c. water and boil. Filter through an ashless filter and wash free from chlorides. Dry, ignite and weigh as SiO_2, doubling the weight to obtain percentage.

In one of the Denver custom laboratories the method is still more exact, special precaution being taken to prevent the loss of soluble silica. The weighed sample is treated with aqua regia and evaporated to dryness. A second similar treatment is given, and as the evaporation nears completion, a little HCl is added. When finally dehydrated, the residue is treated with dilute HCl, boiled and filtered. The filtrate is retained for further treatment. The insoluble thus obtained is fused in a platinum crucible with a mixture of sodium and potassium carbonate, cooled, dissolved in dilute HCl and twice evaporated to dryness. The residue, which is SiO_2 is treated with hot water and HCl, boiled and filtered, combining the filtrate with that first obtained. The combined filtrate is now evaporated to dryness, taken up in dilute HCl and filtered. Additional SiO_2 to the extent of from 1 to 2 per cent is thus obtained, and is added to the larger quantity, ignited and weighed.

Assay for Silver

At the Ohio & Colorado smelter the nitre-crucible assay is used, taking ½ A. T. of the residue in a 20 g. crucible. The fluxing mixture is 20 g. soda, 4 g. silica, 35 g. litharge, and an amount of nitre determined from the following procedure: Determine the carbon in 1 g. of the residue by igniting in an assay muffle and weighing the ash, and from this figure determine the carbon in ½ A. T. Compute the amount of nitre necessary to oxidize this amount of carbon to CO, and for purposes of the assay use this calculated amount of nitre less 2 g. It amounts usually to from 9 to 12 g. The fluxing mixture and residue are thoroughly mixed, covered with 10 g. soda and a teaspoonful of borax glass. Fuse slowly at a low heat for about half an hour, as the assay will boil over if the initial heat is too high. After the charge has settled quietly in the crucible, raise the heat as high as possible and pour. The resulting button is cupelled in the usual manner.

A very satisfactory method is one used at some zinc smelters and by certain custom assayers. The flux contains nitre in amount sufficient, as determined by experience, to give the right size lead button when ¼ A. T. of the residue is used. As the amount of carbon varies in the residue, it may be necessary to make an occasional preliminary assay and vary the amount of nitre according to the result obtained. Little or no trouble is experienced from boiling, and the method is reported to give entire satisfaction.

Purification of Water by Ozone.—To correct a misprint in our issue of Feb. 1, page 160, it should be stated that the cost of purification as carried out by the Baltimore County Water & Electric Company at the present time is less than $2.00 per 1,000,000 gal. of water treated. Besides the aluminium electrodes and micanite dielectric tubes a feature of particular interest in the system is the aspirator tubes used for drawing the ozone through the generators and mixing the gas with the water. The system was developed by the chief engineer of the company, Mr. A. E. Walden. Mr. S. T. Powell presented an interesting paper on this system at the recent Baltimore meeting of the American Institute of Chemical Engineers.

The Use of Low-Grade Phosphates*

BY JAMES A. BARR

When phosphate mining operations first commenced in Tennessee the loss of both high- and low-grade material was large, because of the crude hand methods employed. Practically all rock smaller than 2 in. was thrown back into the pits and covered up with the overburden of subsequent openings.

With the advent of mechanical washers and other mining machinery, together with the abolishing of the contract system, the waste has been steadily diminished, until at present only the finest sand is being rejected; and even this is not lost, for in the majority of cases the water, slime and sand from the washers are run into large settling ponds where practically all of the phosphate-bearing material is caught and stored for future use, or until it is profitable to ship lower grades.

The latest equipment used to recover the fine and lower grades of phosphate sand includes Dorr thickeners, settling tanks, Dorr classifiers and settling ponds served by locomotive cranes with clam-shell buckets, making it possible to save material averaging as fine as 200 mesh. To attempt saving finer material would necessitate a different type of drier, which might be patterned after the multiple-deck roasters.

The present tailing runs from 15 to 25 per cent of available phosphate, which, however, as mentioned before, does not represent a total loss to possible future operations. The demand of the consumer is the greatest factor in regulating the recovery at present.

In the mining of blue rock, which does not require washing treatment, the main waste material is the capping of low-grade kidney formation, generally used for backfilling of the rooms.

By reason of the introduction of improved methods in the early stages and the re-mining of territory skimmed over at the start, at the present time the net result of all operations is a comparatively small loss.

One of the largest temporary losses in the industry occurs in the Florida pebble district, where the recovery of phosphate runs about 25 per cent. The loss is classed as temporary because most of the tailing has been collected in ponds and piles, where it may be reclaimed in the future should developments or improved processes warrant.

In cleaning the Florida pebble phosphate, aside from the elutriation of the clayey binder and strata that sometimes contaminate the rock, the main operation is that of sizing, by which the large phosphate grains are separated from the smaller silica sand. This results in the loss of all the fine phosphate along with the silica. Owing to the similarity of the two in specific gravity, no mechanical process has been developed as yet to effect a further saving.

PROBLEMS AWAITING SOLUTION

As the problem now stands, the development of some concentration process is required to effect a further recovery of the low-grade phosphate in Tennessee and Florida, where the consumers demand a high-grade product.

Taking 68 per cent tricalcium phosphate as the present minimum average grade for Florida pebble and Tennessee blue rock, and 72 per cent for the Tennessee brown rock, it will be seen that the development of such a concentration process is a much needed step in the conservation of the phosphate resources, for by its introduction the commercial limits for use in complete fertilizers could be reduced to 60 per cent and possibly 50 per cent tricalcium phosphate.

In the event of low grades of phosphate becoming commercial products, 20,000,000 tons would become available within a 50-mile radius of Mount Pleasant, Tenn., alone, and perhaps 100,000,000 tons of rock, especially if phosphatic limestone is taken into account. These figures are not claimed to be precise, but are given to indicate that the available phosphatic resources have, so to speak, merely been scratched.

It would seem that the most feasible means for making low-grade phosphate commercially available lies in the development of a smelting or electrometallurgical process whereby the phosphorus content of the rock would be rendered available (soluble) directly and without the introduction of a diluent, as in the manufacture of acid phosphate by the sulphuric acid method. By such a direct conversion, a 60-per cent rock would serve the same purpose as a 72 to 75-per cent rock with the sulphuric acid method.

Along these same lines a combination process may be developed whereby low-grade phosphate and an insoluble potash feldspar or similar potash mineral may, by heat or other treatment, be rendered available for use as a commercial fertilizer. The European war has brought out the importance of such a discovery.

Again, there might be developed another combination process whereby ammonia phosphate could be made directly, as cyanamid is now manufactured with the aid of an electric furnace.

DIRECT USE OF PHOSPHATE ROCK AS FERTILIZER

The greatest possibility of using low-grade phosphate lies in the direct application of finely ground rock to the soil in the raw state, allowing the acids of nature to convert it into soluble forms. One authority on farm fertilizers has said that in the majority of cases a mixture of raw ground phosphate with stable manure makes one of the best fertilizers.

The difficulty with the ground phosphate trade has been that the fineness of grinding was not such that a sufficient percentage of the commercial product became available within any reasonable time. When the grinding industry was new, the fineness was generally 90 per cent through 80 mesh, while now it is 90 per cent through 100 mesh, or even finer. The impalpable powder gives effective and immediate results; therefore the future extension of this particular phase of the phosphate industry will be aided greatly by finer grinding.

Another available source of phosphorus is basic slag from steel manufacture. The United States Steel Corporation has erected a plant in Alabama to prepare such slag for fertilizing purposes and has placed the product on the market.

This form of fertilizer has been successfully used in Germany.

An average grain crop requires and takes about 15 lb. of phosphorus per acre from the soil in which it is grown. This phosphorus must eventually be replenished, for with a meager amount of phosphorus in the soil the crops are poor, and with none the harvest is nil. The majority of farmers fail to replenish the soil with the elements taken out by the crops, and at no very distant time they will be drawing heavily on our phosphate resources.

In view of this, the use of the lower grades is certain to become common.

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*A paper to be presented at the New York meeting (February 14 to 17) of the American Institute of Mining Engineers.

Lead and Zinc Smelter to Reopen.—The lead and zinc smelter in Webb City, Mo., will be reopened shortly and will be operated by the Picher Lead Co. of Joplin. This smelter has been idle for several years and when in operation will employ about 100 men.

The Hydrometallurgical Treatment of Complex Gold and Silver Ores

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BY G. H. CLEVENGER

The term *"complex ore"* has never been clearly defined, nor is it probable that it ever will be. If defined upon the basis of the ability to obtain a high extraction, an ore which could be treated at a profit and a high extraction realized could be called a *simple ore*, no matter how complicated the process for realizing this end might be.

Then, again, the classification might be based upon the nature of the process necessary for the satisfactory treatment of the ore; thus an ore which required a complex process or combination of processes might be classified as a *complex ore*, while an ore with which a simple process sufficed might be called a *simple ore*. The latter suggestion seems to be the more logical basis for a classification. But it would be difficult if not quite impossible to draw the line as to where a process ceased to be simple and where it became complex, as this would be largely a matter of personal opinion upon which there could be no general agreement.

Despite the difficulty of satisfactorily defining a *"complex ore"* there is a constant effort being made to treat ores by hydrometallurgical methods which have previously resisted such attempts, and metallurgists are also constantly endeavoring to develop more simple and direct processes of treatment which will either result in a higher extraction, lower working costs, or both.

Historical

The present status of the hydrometallurgy of gold and silver is perhaps best understood by going back to the beginning of hydrometallurgical practice and briefly tracing the various developments down to the present day.

The oldest hydrometallurgical process, if such it really could be considered, was the simple operation of panning in water a gold-bearing sand or ore which had been crushed between stones. This was soon supplemented by the passage of the finely ground ore with water over the hair side of the skins of wild animals.

The next development was the *early Roman amalgamation process*, first described in Pliny,[1] which was conducted by agitating the pulverized ore with mercury and water in earthen vessels. Since this process was only effective for the recovery of metallic gold or silver from simple ores it was but natural that the *early European amalgamation process*,[2] capable of treating more complex ores, particularly those of silver, should be developed. The prominent features of this process as ordinarily carried on was the grinding of the ore in a stone mill with water, mercury, salt, and some compound of copper. At times other reagents were added, but they were not essential. Although this process was in use in various parts of Europe at a much earlier date, it was first *described by Biringuccio*[3] in 1540.

Bartolme Medina introduced the *Patio*,[4, 5] or, as it was sometimes known, *"Mexican Amalgamation Process,"* at Pachuca, Mexico, in 1557. The finely ground ore, moistened with water, was mixed with mercury, copper sulphate and salt on large stone-paved areas (patios) by the treading of horses or mules. Since all the mercury at this time came from Europe, it seems probable that Medina had as a basis for the development of this process a general knowledge of the European process of amalgamation.

Be this as it may, great credit was due him for this new and novel metallurgical process which made possible for the first time the treatment of large lots of ore by a hydrometallurgical process at a reasonable working cost. So thoroughly did he adapt the general principles of amalgamation to the special conditions met with in Mexico that few improvements were ever made in the original process, which held the field for the treatment of silver-gold ores in Mexico until within recent years, when the cyanide process, capable of giving a higher extraction of the gold and silver, was developed to the point where it could be more economically applied than the Patio process.

The *Cazo* or *Fondo process* was invented by the priest Barba,[6] residing in Bolivia, while experimenting as an alchemist in 1609. This process comprised stirring the finely ground ore with water, mercury and salt in a copper kettle heated over a fireplace. Until about 1793 the Fondo process was used in South America and Mexico with but few variations from the method as originally described by Barba. At about this time a larger vessel having a copper bottom and wooden sides, in which the stirring was done by mule power, came into use. This was known as the *Fondon process*.[7]

The *Tina process*,[8] employing an iron-bottomed wooden vessel, in which the ore was ground in water and amalgamated with mercury, was used at Kongsberg, Norway, for treating native silver ores. A modification of the Norwegian Tina process[9] was also used at a later date in Chile for treating simple ores.

The chloridizing roast for silver ores, introduced by Born,[10] was first combined with the Cazo process[11] at Schemitz, Hungary, in 1786. Barrel amalgamation[12] in combination with the chloridizing roast was first introduced at Halshüche Hütte, near Freiberg, in 1790.

The *Washoe process*,[13] developed through the combination of the California stamp mill with an elaboration of the Norwegian Tina for fine grinding and amalgamation (pan) and the chemicals of the Patio process, was introduced for the treatment of Comstock ores in 1860. This amalgamation process, adapted to American conditions, rapidly assumed the same position in the United States for the treatment of silver-gold ores as was occupied by the Patio process in Mexico for the treatment of the same class of ores.

A later development when treating complex ores was to give a chloridizing roast preceding grinding and amalgamation in the pans. This was known as the *Reese River process*.[14]

The *Kröhnke*[15] *process of barrel amalgamation* was introduced into Chile in 1862, but had a limited use. The date of the first use of amalgamated copper plates

[1] Naturalis Historiae. Lib. XXX. Cap. VI, sect, 32. Translation—Metallurgy of Silver and Gold, John Percy, London, 1880, p. 559.

[2] De Re Metallica, Georgius Agricola. Hoover's translation from the first Latin edition of 1556. Book VIII, pp. 295, 298.

[3] De la Pirotechnia, per Vanoccio Biringuccio, Sennese, Ventia, 1540, Lib. IX, Cap, XI, fol. 142. Translation—Metallurgy of Silver and Gold, John Percy, London, 1880, pp. 560, 561.

[4] Mexico. H. G. Ward, London, 1829, pp. 196, 199.

[5] The Patio Process for Amalgamation of Silver Ores, by Manuel Valerio Oretega. Trans. Amer. Inst. Min. Engrs., Vol. XXXII, pp. 276, 285.

[6] Arte de los Metales, antes cit. lib. III, cap. 16. Translation, Metallurgy of Silver and Gold, John Percy, London, 1880, p. 563.

[7] Laur, Annales des Mines, 1871, Sixth series, pp. 216-236. Also Metallurgy of Silver and Gold, John Percy, London, 1880, pp. 661, 665.

[8] Gründlicher Unterricht, Schlüter, pp. 122 et seq. Translation, Metallurgy of Silver and Gold. John Percy, London, 1880, pp. 565-567.

[9] Metallurgy of Silver and Gold. John Percy, London, 1880, pp. 567-573.

[10] Handbook of Metallurgy. Carl Schnabel. Translation by Henry Louis, London, 1898, p. 688.

[11] Loc. cit.

[12] Ibid. p. 689.

[13] U. S. Geological Exploration of the Fortieth Parallel. Mining Industry, James D. Hague. Treatment of Comstock Ores, pp. 194, 272. Chemistry of the Washoe Process, pp. 273, 293.

[14] The Reese River Process, The Mettacom Mill. R. W. Raymond. Mining Statistics West of the Rocky Mountains (1870), pp. 733, 740.

[15] Metallurgy of Silver, by H. F. Collins, London, 1900, pp. 69-72.

as now used for the treatment of simple gold ores is not definitely known, but it seems probable that it began by supplying the inside of the mortar with copper plates, suggested, no doubt, by the use of the copper kettle in the Fondo process; this was followed by the use of plates outside of the mortar. Outside plates were in use as early as 1856[16] at Grass Valley, California. Previous to this the metallic gold was collected upon the hairy side of the skins of animals, blankets or other rough surfaces.

The *Batopilas*[17] *process* of amalgamation in pans with cyanide solution was described in 1911. The *Nipissing*[18] *process* of amalgamation in a silex-lined tube mill with very strong cyanide solution was introduced early in 1911. The processes so far considered were all amalgamation processes. In 1848 *the Plattner chlorination process,*[19] the earliest lixiviation process, was introduced. In this process an oxidizing roast of the ore preceded its treatment with chlorine gas. Through the suggestions and work of a number of men, the more modern process of *barrel chlorination*[20] was evolved in 1890. This process was more suitable for the treatment of large tonnages of ore than the Plattner process. *Vat chlorination*[21] was the latest development in gold processes employing chlorine as a solvent.

A silver lixiviation process based upon the chloridizing roast of the ore followed by leaching with a solution of sodium hyposulphite was introduced in 1858 by von Patera.[22] This was followed in 1860 by the proposal of *Kiss*[23] to use the calcium salt rather than the sodium salt on account of the supposed greater solubility of gold in calcium hyposulphite. In 1884, *Russell*[24] introduced the use of an extra solution of sodium-copper hyposulphite and sodium carbonate for precipitating the lead.

The *Augustin*[25] and the *Ziervogel*[26] processes, introduced in the 40's, were used in a limited way almost exclusively for the treatment of matte. However, the recent application of the Augustin process[27] in connection with blast roasting for the treatment of low-grade complex silver ores containing lead and copper is worthy of note.

The first proposal for the treatment of gold and silver ores by cyanide solutions was made in 1867 by *Ray,*[28] although *Elkington*[29] had found in 1840 that gold was soluble in cyanide solutions with the aid of electricity, and *Bagration*[30] had discovered in 1842 that gold was soluble in cyanide solutions without the use of electricity. *MacArthur and Forrest,* in 1886,[31] developed a process employing dilute solutions of cyanide, and zinc precipitation. About the same time *Siemens*[32] developed a process employing very dilute cyanide solutions and electrolytic precipitation.

It has required almost thirty years and the work of many men to bring the cyanide process to its present high state of perfection. Originally the cyanide process was only considered generally applicable to the treatment of simple gold ores and on account of the lack of suitable means for treating the slime high extractions were the exception rather than the rule, so that the cyanide process did not at once become a serious competitor of the older processes. The Plattner chlorination process for limited amounts of high-grade gold concentrate and the barrel chlorination process for large tonnages of siliceous gold ores continued to hold the field for the treatment of complex gold ores, and plate amalgamation with or without concentration for simple gold ores. The Patio amalgamation process in Mexico, and the Washoe amalgamation process, and to a less extent the hyposulphite lixiviation processes in the United States, continued to hold the field for treating silver-gold ores.

Cyanidation was first used to supplement the recovery made from gold ores by plate amalgamation. With the introduction of fine grinding and the various types of slime filters, it became a formidable rival of the older processes of amalgamation and chlorination for the extraction of both gold and silver. As the operation of the process became better understood, it gradually displaced the Patio process in Mexico, and the Washoe and chlorination processes in the United States. To-day all of the older processes of amalgamation, with the exception of plate amalgamation and the Nipissing process, are obsolete. The same may be said of chlorination, which made its last stand in treating the Cripple Creek telluride ores. The hyposulphite silver leaching processes long since became obsolete without ever becoming an important factor in ore treatment.

The cyanide process, with the exception of plate amalgamation and the Nipissing process, therefore, at the present time entirely holds the field for the hydrometallurgical treatment of gold and silver ores. For the purpose of discussion in this paper, amalgamation is considered an accessory operation. The advantage of treating both gold and silver ores by the same process is apparent. Silver was not recovered in the chlorination process and, if present to an appreciable extent, interfered with the gold extraction through the formation of insoluble silver chloride. Attempts to recover the silver by a subsequent treatment with sodium hyposulphite solution did not prove satisfactory. The silver amalgamation and lixiviation processes did not recover a high percentage of the gold and left much to be desired in the recovery of the silver. Unfortunately, gold and silver do not occur in ores in the sharply defined lines drawn at times by the limitations imposed by certain processes of ore treatment. The cyanide process being both a gold and a silver process, has the necessary flexibility regarding the recovery of the two metals from the same ore.

Preliminary or Accessory Operations

In order to properly consider the preliminary or accessory processes or operations which are available for use in conjunction with the cyanide process for increasing its effectiveness, it will be necessary to return to certain of the earlier processes and consider the function of certain of these operations as carried on in them.

The oxidizing roast was first applied as a preliminary operation with the old Plattner chlorination process for

[16] The writer has corresponded with R. W. Raymond, the late S. B. Cristy, and many others familiar with early metallurgical development in the United States, in a vain endeavor to ascertain where and by whom amalgamated copper plates were first used in the recovery of gold. Mr. W. L. Oliver has informed me that amalgamated copper plates were in use as early as 1856 at Grass Valley, California, U. S. A.

[17] The Milling and Batopilas Native Silver Ores. Walter M. Brodie. The Mexican Mining Journal, Vol. XII, pp. 21, 23.

[18] Nipissing High Grade Mill. R. B. Watson. Engineering and Mining Journal, Vol. 94, pp. 1077, 1080.

[19] Metallurgy of Gold, by T. R. Rose, London, 1906, pp. 239, 256.

[20] The Present Development of the Barrel Chlorination Process, by John E. Rothwell. Mineral Industry, Vol. V, pp. 261, 284.

[21] The Chlorination of Gold Ores at Mount Morgan, Queensland. E. W. Nardin. Proc. Inst. Civil Engrs., Vol CXLII (1900), pp. 297, 307.

[22] Handbook of Metallurgy, by Carl Schnabel, Translation by Henry Louis, London, 1898, pp. 718, 733.

[23] Ibid., pp. 732, 733.

[24] The Lixiviation of Silver Ores with Hyposulphite Solution, by Carl A. Stetefeldt (Second Edition). The whole work deals with the Russell process, or closely related matter.

[25] Handbook of Metallurgy, by Carl Schnabel. Translation by Henry Louis, London, 1898, pp. 708, 718.

[26] Ibid., pp. 740, 749.

[27] Chloridizing Leaching at Park City, Utah, by Theodore P. Holt. Trans. Amer. Inst. Min. Engrs., Vol. XLIX, pp. 183, 197.

[28] U. S. Patent No. 61,866, 1867.

[29] British Patent No. 8447, 1840.

[30] On the Property which Potassium Cyanide and Ferrocyanide Possess of Dissolving Metals, by Prince Pierre Bagration. Bul. de l'Acad. Impl. de St. Pet. (1843), II, p. 136.

[31] British Patent 14,174, 1887.

[32] Electrical Precipitation of Gold, by A. von Gernet. Proc. Chem. and Met. Soc. of Africa, Vol. I, pp. 28, 30.

gold. Roasting was absolutely essential to the successful operation of all the chlorination processes. The oxidizing roast should never be used in conjunction with any process for the recovery of silver, on account of the fact that it renders a large proportion of the silver refractory to all known hydrometallurgical processes, including the cyanide process.

The reactions occurring during roasting and the final chemical combination, or physical change of the silver existing in an ore at the end of the operation, have never been determined. This is a matter which is well worth careful investigation, as it appears that we have the same phenomenon to deal with in the treatment of certain naturally occurring oxidized ores of silver, particularly those containing oxidized manganese minerals. My own researches have led me to believe that the silver is not in combination with manganese, but that probably the oxides of manganese have exerted an influence upon the occurrence of the silver through their behavior as oxidizing agents. Certain silver ores containing the oxides of manganese are not refractory to cyanide treatment, but this fact in no way controverts the above theory.

The oxidizing roast as practised with the chlorination process is still employed in conjunction with the cyanidation of certain of the higher-grade sulphide and sulpho-telluride gold ores. It is significant that but little of the small proportion of silver which these ores generally contain is recovered during cyanidation. Several hundred tests made under the writer's direction have indicated that the raw sulpho-telluride ores (containing approximately 1.0 oz. of gold per ton) of the Cripple Creek district, with certain exceptions, will yield almost as high an extraction when ground very fine and given a long period of agitation as when treated by the common practice of roasting, followed by regrinding and cyanidation. The raw ore under the above conditions also yields a quite good extraction of the silver which is invariably below 1.0 oz. per ton. Notwithstanding this fact, the roasting process so far as we now know, under the conditions of cheap fuel and a roasting cost of 50 to 60 cents per ton, still holds the field for the following reasons: It is not necessary to grind so fine; the extraction is a matter of hours rather than days; the cyanide consumption is less and the solutions are less likely to become foul. However, investigation is constantly going forward in this direction, and it would not be surprising if raw treatment would eventually win out.

A recent development in the roasting of ores preliminary to hydro-metallurgical treatment is the application of blast roasting or continuous suction roasting machines of the Dwight-Lloyd type. This bids fair in certain cases to meet all the requirements of roasting and to reduce the cost of the operation materially.

The thoroughness of the roast determined whether the results were good or bad in the chlorination process and, while this is less true in the cyanide process, yet proper roasting is of prime importance if the best results are to be realized. It seems probable that careful investigation of the whole problem of roasting would eliminate the irregularities which sometimes arise. The oxidizing roast is only applicable to gold ores, and then only when the results are sufficiently improved, as regards extraction or costs, to justify its use.

The chloridizing roast, introducing salt during the roast, was first applied in connection with the Fondon process, and later became quite generally used with the barrel amalgamation process, the Washoe process and the silver lixiviation processes for treating refractory ores. The chloridizing roast has never been favored for gold ores on account of the high volatilization losses; in fact, this is also its weakest point with silver, since silver chloride is quite volatile. It has been used at times in conjunction with the cyanide process, but its use in this connection has entirely passed out. It is claimed that volatilization of the silver is entirely under control in the newly-proposed system of blast and suction roasting. One possible application of the chloridizing roast under favorable conditions is in connection with the treatment of the so-called refractory manganese-silver ores, as most of the silver is converted into readily soluble silver chloride.

The losses of gold and silver occurring during roasting in a well-regulated plant have never been accurately determined, and, indeed, it would be a most difficult thing to do. Although such losses may be small, they constitute a factor which cannot be ignored.

Water concentration was applied in conjunction with many of the earlier processes, as, for example, the Patio and Washoe processes. It was early introduced, following simple plate amalgamation, for the purpose of augmenting the recovery from gold ores. Eventually all the tailing from the modern barrel chlorination plants were concentrated before being discarded. Concentration is a valuable adjunct of the cyanide process and has been practised both when crushing in water and when crushing in solution.

Generally the operation precedes cyanide treatment, on account of being more conveniently applied at this point, but in many cases it would be advantageous if it were feasible to have it follow cyanide treatment. When it precedes cyanide treatment, a considerable proportion of the readily soluble gold and silver may be removed in the concentrate, when it might be more advantageous to recover them as bullion. In the case of low freight rates and favorable smelter contracts this may be an advantage rather than otherwise; but for the majority of plants the more gold and silver turned out as bullion the better. Therefore, the ideal practice in most cases would be to recover all the gold and silver possible as bullion by the cyanide process and then concentrate the tailing to recover as high a percentage of the remaning gold and silver which had escaped dissolution as feasible. Obviously there are many cases where the percentage of extraction of gold and silver from the residue leaves no margin for concentration.

At times concentration can be introduced into the cyanide flow sheet to advantage for the purpose of removing the minerals which are difficult, if not impossible, to treat by the regular milling scheme, so that they may receive the special treatment necessary without having to incur the expense of subjecting the whole tonnage to the special treatment made necessary by a comparatively small proportion of refractory minerals. On the other hand, the introduction of concentration adds to the first cost of the plant and the complexity of its operation, so that these disadvantages must be carefully weighed against the advantages which are likely to accrue.

Chemical reduction is one of the latest developments in preliminary treatment and was, therefore, not used in conjunction with any of the earlier processes. Such a process is most feasible when carried on in an alkaline solution, as acids attack the gangue of most ores. Reduction may be accomplished through the use of dilute caustic soda solution with aluminium or zinc, or through electrolysis in a suitable solution by bringing the ore to be reduced into intimate contact with the cathode.

The only commercial application of reduction is at the Nipissing low-grade mill, where caustic soda and aluminium are used for converting silver sulphide and sulpho-antimonide minerals into metallic silver prior to

cyanidation. This preliminary treatment lessens the period of agitation necessary with cyanide solution.

Many experiments involving the preliminary reduction of telluride gold ores in caustic soda solution with aluminium and zinc, as well as cathodic reduction in both caustic soda and salt solution, have shown that with cheap fuel and efficient roasting this form of preliminary treatment would present no advantage. The gold telluride compounds are in general more difficult to reduce than the sulphide and sulpho-antimonide silver minerals.

Amalgamation, particularly plate amalgamation, is an important preliminary treatment with gold ores which contain any considerable proportion of readily amalgamable gold. In fact, in certain cases where a considerable proportion of the gold occurs native, the bulk of the extraction may be effected by amalgamation. However, the cyanide process is necessary for recovering the portion not recovered by amalgamation.

There is considerable difference of opinion among metallurgists regarding the extent to which amalgamation should be applied in hydro-metallurgical work. Those having the extreme point of view maintain that with sufficiently fine grinding to bring all the gold or silver to the degree of subdivision necessary for rapid dissolution, amalgamation may be dispensed with and the whole of the gold recovered by cyanidation. This can be done in certain cases, even with silver ores where a considerable weight of metal is involved, but it means a higher cost for grinding and generally also a higher cost for cyanide treatment and, therefore, may not be so profitable as coarser grinding and amalgamation followed by cyanidation.

Special types of amalgamation, such as the tube mill amalgamation practised at the Nipissing mine at Cobalt, Ontario, Canada, becomes absolutely necessary in the case of the very high-grade silver ores treated in this district which contain a large proportion of native silver, or closely allied minerals.

Flotation, the most recent development in ore dressing, which has become such an important factor in the recovery of the rare metals from their ores, is becoming a valuable adjunct to the cyanide process in connection with the treatment of the precious metal ores. Recently very good experimental results have been obtained upon refractory manganese-silver ores by giving a preliminary sulphidizing treatment followed by flotation. Flotation in alkaline solutions is also a recent development which promises much.

The data now available indicate that flotation may be used prior to cyanidation for removing minerals interfering with cyanide treatment, such as the sulphide copper minerals or such minerals as the tellurides and certain silver minerals which might be more advantageously removed in this way than decomposed and dissolved by cyanide. In most cases the tailing from flotation appears to be amenable to cyanide treatment, but the concentrate is not, even though there be no refractory mineral present. Gold concentrate is amenable to cyanidation after roasting, but silver concentrate is not on account of the refractory combination of silver formed by the oxidizing roast. Metallurgists are at work upon this problem and it is quite possible that a solution will be found in the near future.

The treatment of the tailing from cyanidation is a most attractive field, particularly since flotation in alkaline solutions has become possible. So far flotation has not been accomplished in cyanide solutions. Another very attractive field is the direct treatment of very low-grade ores which would not pay to cyanide. In many such cases a high extraction could be obtained, the tailing rejected and the concentrate cyanided or shipped. This proposal is, of course, made upon the assumption that certain of the difficulties which now stand in the way of treating flotation concentrate will be overcome.

Important Factors in the Operation of the Cyanide Process

Inasmuch as the cyanide process is of vital importance in the present-day hydrometallurgy of gold and silver, a discussion of the more important controlling factors of the process is in order.

The degree of comminution of the ore is one of the most important factors, particularly in the treatment of silver ores. The purpose of grinding is to free the gold and silver minerals from the inclosing gangue and to reduce the mineral particles to such a size that they are readily dissolved. Grinding should be carried on in such a manner as to waste as little power as possible in grinding the worthless gangue and still fulfill the above named conditions to the fullest extent possible, since the less the mineral is protected by the gangue and the greater the surface of the mineral exposed to the solution, the more rapid the rate of dissolution.

It then follows, for example, in the case of certain silver ores, that while fine grinding may not produce any greater ultimate extraction, yet in general it will materially reduce the time of treatment necessary. But this advantage is not always realized without the disadvantage of greater consumption of cyanide arising, since finer grinding not only causes a greater surface of the minerals containing the precious metals to be exposed to the solution, but also a greater surface of those minerals, if present, which may act as cyanicides. Careful correlation of the time of treatment with degree of comminution may serve to minimize this difficulty.

Selective grinding whereby the heavy mineral particles are ground finer than the lighter particles of gangue takes place automatically to a greater or less degree in closed circuits where hydraulic or mechanical classifiers are employed so that actually in the majority of plants the heavy mineral particles are ground finer than the gangue. The additional cost of finer grinding, together with the attendant disadvantages which may arise, must in each case be carefully weighed against the additional extraction possible, or the decreased time of treatment necessary to obtain a given extraction.

The concentration of the solution of cyanide is a vital point. In general, the stronger the solution, within certain fairly well defined limits, the more rapid the dissolution, and, furthermore, the less the interference of a given percentage of impurity which might be present in solution. On the other hand, a greater proportion of impurity may be dissolved by the stronger solution. In the case of a plant which is operating at forced capacity a stronger solution may be used to advantage in order that the maximum extraction may be attained under this condition of operation. Stronger solutions may result in increased cyanide consumption, although the magnitude of this loss in a properly operated plant is not so great as has been supposed.

However, in a plant where there is considerable mechanical loss of solution the additional cyanide loss would prove an important factor. For this reason a strong solution is not generally favored when continuous decantation is used. The tendency in some cyanide plants has been to use a lower concentration in cyanide than that capable of giving the highest economical result. This probably is through the fear of excessive cyanide loss.

The alkalinity both as regards the kind of alkali and its concentration is perhaps only second to the concen-

tration of the solution in cyanide in importance. The effective alkalies are the alkaline hydroxides, the most available being caustic soda and lime. Lime in contact with the solution becomes calcium hydroxide, slightly soluble, but sufficiently soluble to be effective as an alkali in cyanide treatment. Lime in the majority of localities is much cheaper than caustic soda and, fortunately, is in the majority of cases the most satisfactory alkali to use. Many of the impurities which form soluble sodium salts form with calcium insoluble compounds which are eliminated with the tailing, thereby maintaining the solution at a higher degree of efficiency. There is evidence to indicate that when certain antimonial minerals are cyanided caustic soda may prove the more satisfactory alkali; this, however, is not definitely proved, and most certainly is not true in all cases.

The concentration of alkali, or as it is usually termed the protective alkali, may vary from practically 0 to 100 points.* At the Homestake a very low alkalinity is conducive to the best extraction, while in treating raw Cripple Creek ores a rather high alkalinity, employing lime, results in the highest extraction and the lowest consumption of cyanide.

Dilution or ratio of solution to dry ore in the pulp undergoing treatment is generally recognized as a factor to which proper attention must be paid if the highest extraction is to result. If a higher dilution is used than is necessary the capacity of the plant is cut down, and, on the other hand, if the dilution be too low the maximum extraction is not attained. It is quite impossible to give general figures for minimum dilution as these figures vary for different ores. In general, lower dilutions can be used with gold ores than with silver ores, probably on account of the greater weight of metal to be dissolved in the latter case. Other conditions being equal, it appears that lower dilutions can be used with the Pachuca agitator than with the mechanical agitator on account of the greater proportion of air brought in intimate contact with the pulp.

The time of treatment depends upon the character of the silver and gold minerals and upon their degree of comminution, and, as previously pointed out, also upon the concentration or strength of the solution in cyanide. It therefore follows that finer grinding, or increase of the cyanide concentration of the solution, or both, will in general result in reducing the time of treatment necessary. In cases where the cost of power is high, and as a consequence the cost of fine grinding would be excessive, the ore may be ground to the point where the minerals are liberated from the gangue and then separated into sand and slime, the slime being treated by agitation followed by either decantation or filtration. The sand containing the coarse mineral particles requiring a long period of contact with the solution for dissolution can be given the long period of contact necessary at a reasonable cost by leaching. Cases of this kind clearly indicate where combined sand and slime treatment can be employed to advantage.

The method of precipitation employed, assuming equal efficiency as regards the precipitation and recovery from solution of the precious metals, may exert an important influence upon extraction, either through introduction of the precipitant used into solution or its failure to precipitate certain interferring elements. For example, zinc in the presence of arsenic may interfere in the treatment of certain ores. If aluminium precipitation is used the difficulty is overcome through the elimination of zinc. When copper reaches a certain concentration in solution difficulties arise with both extraction and precipitation. Neither zinc nor aluminium precipitate copper to any extent, hence if copper is to be removed from solution electrolytic precipitation must be used.

Carbon, which at times occurs in gold and silver ores, may occasion difficulty in cyanidation. It has been generally assumed that carbon occurs in gold and silver ores in the form of graphite, but the evidence available by no means supports this view in all cases. The two extremes of carbon as regards its behavior in cyanide solutions are graphite and charcoal. Graphite is dense and does not possess pores, therefore cannot occlude gases, while charcoal is porous and has the property of occluding relatively large volumes of gases. Graphite does not precipitate gold and silver from cyanide solutions while charcoal does. Intermediate between these two extremes are various forms of carbon which will precipitate gold and silver to a greater or less extent.

Feldtmann[28] has shown that the graphitic or carbonaceous schist from the mines of West Africa will precipitate gold from cyanide solutions roughly in proportion to the carbon content or the schist. Gold so precipitated is not soluble to any appreciable extent in fresh cyanide solution, but is soluble in sodium sulphide solution. He thinks that the compound formed is possibly carbonyl aurocyanide, which may react with sodium sulphide according to the below equation:

$$AuCNCO(CN)_2 + Na_2S = NaAu(CN)_2 + NaCNS + CO$$

The only gold ore containing carbon which I have had an opportunity of investigating carefully showed but little tendency to cause premature precipitation. This fact, together with its general appearance, led to the conclusion that the carbon in this ore was largely graphite. However, the graphite was found to definitely interfere with cyanidation. Experiments with defflocculation of the colloidal portion of the ore including the graphite gave negative results, but it was discovered that if the physical state of the graphite were altered, most conveniently by heating, that the extraction was wonderfully improved. The results in Table I show clearly the effect of heating.

TABLE I

	Per Cent of Gold Extracted	Pounds of Cyanide Consumed	Per Cent of Graphite
Wet slime	36.14	0.9	1.65
Dried at 100 deg. C.	64.60	0.6	...
Check	64.60	0.6	...
Heated to 300 deg. C.	80	0.7	1.55
Check	83.00	0.5	...
Wet slime boiled with water, graphite skimmed off	56.00	0.6	...
Wet slime boiled with water, graphite not skimmed off	54.60	1.32	...
Roasted in open dish	90.10	0.12	...
Roasted in open dish	90.10	0.24	0.63

Practically all gold or silver ores can be treated by the cyanide process or by the cyanide process in combination with some other preliminary or accessory treatment, with the exception perhaps of certain ores containing a considerable amount of copper, lead, etc. In such cases, the desirability of recovering the base metals leads to smelting the ore upon either the lead or copper basis, and the necessity for hydrometallurgical treatment for the recovery of the gold and silver disappears since the precious metals are recovered by the smelting operation in connection with the base metals. Despite the fact that most ores can be successfully treated by the skillful application of the cyanide process, either alone or in combination with other accessory operations, it cannot be said that such treatment is

*The writer has long used a system of alkalinity numbers, similar to the system proposed by Sherwood, except that instead of using a n/5 solution of acid, an empirical solution (in the case of oxalic acid 14.6071 grams per liter) is used. When a 50 c.c. sample of solution is taken for titration, each 0.1 c.c. of the acid required corresponds to 1 point of alkalinity; 100 points correspond to a saturated solution of lime in water at ordinary temperatures.

[28] Precipitating Action of Carbon in Cyanide Solutions. W. R. Feldtmann. Trans. Inst. Min. and Met., London, April, 1915.

always profitable, so that much remains to be done in the way of simplifying and lowering the operating costs of hydrometallurgical processes for the treatment of gold and silver ores, and in many cases improving the percentage of extraction.

Exemplification of Present-Day Adaptation of the Cyanide Process.—A clearer understanding of the possibilities in connection with the treatment of complex ores will perhaps be gained from a consideration of the solution of certain ore treatment problems which are exemplified by some of the adaptations of cyanidation in use at present for the treatment of a great variety of ores. On account of the high percentage of the precious metals recovered, the ores treated may not be looked upon as being complex ores. Certain of the cases which are now cited as an accomplished fact at one time could not be treated, or were treated by a single simple process and a very much lower extraction realized. The metallurgical practice at the older mines has been largely a matter of evolution. For example, the cyanide process has been added to plate amalgamation; it has displaced the chlorination process, and has been combined with concentration for the treatment of a great variety of gold and silver ores.

Example A.—The Homestake mine[34] (Lead, S. D., U. S. A.) produces a low-grade gold ore ($3.59 per ton) in which a considerable proportion of the gold occurs in the metallic state, therefore recoverable by plate amalgamation. However, this ore did not yield a high extraction, according to present standards, by plate amalgamation, although the process was carried to the extreme limit of its possibility through the introduction of an unusually large plate area. This was on account of the occurrence of a portion of the gold intimately associated with pyrite and arseno-pyrite. The recovery by plate amalgamation, the original process, was first supplemented by concentration, then concentration was discontinued and the cyanide process was introduced for treating the whole of the pulp. This recovered a much higher total percentage of the gold and permitted the shipment of the whole product as bullion. The pulp is separated into sand and slime—the sand treated by leaching in vats and the slime by leaching in the Merrill filter press. Table II gives the extraction results.*

Table II. Homestake Mine

	Per Cent
Extraction by amalgamation employing a large plate area	69.0
By amalgamation upon plates after regrinding	0.8
By cyanidation from sand	15.1
By cyanidation from slime	9.7
Total by amalgamation and cyanidation (in 1914)	94.6

Example B.—The Treadwell mine[35 and 36] (Douglas Island, Alaska) produces a lower grade gold ore ($2.85 per ton in 1914) than the Homestake. It is possible to make a considerable recovery of the gold by plate amalgamation. A portion of the gold in this ore occurs intimately associated with pyrite, therefore concentration has from the beginning followed plate amalgamation. The early practice was to roast the concentrate and treat it locally by the Plattner chlorination process; later it was found more economical to ship the concentrate to Pacific Coast smelters. At the present time the concentrate is reground and cyanided locally so that this company ships its whole product as bullion. Unlike the Homestake, the balance of the tailing does not pay to cyanide. Table III gives the extraction results.

Table III. Treadwell Mine

	Per Cent
Extraction by amalgamation employing a moderate plate area	48.6
By concentration	43.6
Total extraction by amalgamation and cyanidation	92.2
By cyanidation from the concentrate	97.2
Total net extraction by amalgamation, concentration and cyanidation of the concentrate	91.0

Example C.—The Cripple Creek ores[37] (Cripple Creek, Col.), of moderate grade (+ 1.00 oz. gold per ton), in which very finely divided native gold and the telluride minerals of gold, and to a slight extent those of silver, occur intimately associated with pyrite and other sulphide minerals, were formerly treated by roasting and barrel chlorination followed by concentration. The present practice is to roast, regrind in cyanide solution, concentrate on woolen blankets, separate into sand and slime, cyanide the sand by leaching and treat the slime either directly by pressure filters or by agitation and vacuum filtration. The blanket concentrate is either melted directly or is treated by amalgamation.

The general principle of blanket concentration is one of the most ancient of gold recovery processes, and therefore an operation which the modern metallurgist is prone to condemn when he first sees it in operation in a modern mill. However, careful investigation increases his respect for this time-honored method of gold recovery when he finds that under the special conditions obtaining in these plants it is performing a most useful function in a most economical manner. Table IV gives the extraction results.

Table IV. Cripple Creek

	Per Cent
Extraction by blanket concentration (approx.)	25.0
By cyanidation (approx.)	70 to 73.0
Total extraction by concentration and cyanidation (approx.)	95 to 98.0

Example D.—The Portland Gold Mining Company's low-grade Cripple Creek ores,[38] generally below 0.15 oz. gold per ton (Cripple Creek, Col.), are of the same general character as the higher grade ore. The ore is crushed in cyanide solution, concentrated, and the sand separated from the slime. The sand is rejected, as it is too low-grade for further treatment, but the slime is cyanided. The concentrate is shipped a short distance to the smelter. Although more profitable under present conditions to ship the concentrate, it could be roasted and cyanided locally, or might possibly be treated by very fine grinding and direct cyanidation. Table V gives extraction results.

Table V. Portland Gold Mining Company

	Per Cent
Extraction by concentration	28.0
By cyanidation from slime	52.0
Total extraction	80.0

Example E.—Goldfield Consolidated mine[39 and 40] (Goldfield, Nevada) formerly produced a high-grade complex gold ore (average November, 1909, to Oct. 31, 1910, —1.785 oz. per ton), but as operations have continued

*The figures given for extraction in the different examples are not in all cases the same as those which have appeared in descriptions of the practice which have previously appeared. This comes about either through the realization of a better extraction on account of improvements in the practice or a decreased extraction through increased complexity of the ore. The brief resumés of practice and figures for extraction given are strictly up to date and authoritative with but one exception. In this connection the writer is indebted to the following well known managers for supplying the necessary information and giving permission for its publication: Mr. Fred Bradley, Mr. Allan J. Clark, Mr. G. M. Taylor, Mr. Thos. B. Crowe, Mr. A. H. Jones, Mr. R. B. Watson, Mr. J. W. Hutchinson, and others.

[34]The Metallurgy of the Homestake Ore, by Allan J. Clark and W. J. Sharwood. Institution of Mining and Metallurgy. Vol. 22, pp. 68-176. Discussion, pp. 176-2-4.

[35]The Treadwell Group of Mines, Douglas Island, Alaska, by Robert A. Kinzie. Trans. Amer. Inst. Min. Engrs. Vol. 34, pp. 334, 386.

[36]The Cyanide Plant at the Treadwell Mines, Alaska, by W. P. Lass. Trans. Amer. Inst. Min. Engrs. Vol. 42, pp. 785-818.

[37]Private notes of the author.

[38]Private notes of the author.

[39]Operation of the Goldfield Consolidated Mill, by J. W. Hutchinson. *Mining and Scientific Press*, Vol. 102, pp. 616-620, 652-654, 686-688, 716-719, 782-784.

[40]Treatment of Concentrate at Goldfield Consolidated Mill, by J. W. Hutchinson. *Mining and Scientific Press*, Vol. 106, pp. 170-172, 204-209.

the grade has become lower. The old practice was to crush in water, amalgamate, concentrate and cyanide. The concentrate was treated by a special process and the residue shipped to the smelter. In the present practice amalgamation has been dispensed with and the concentrate is treated locally by cyanidation after roasting. Table VI summarizes the results.

TABLE VI. GOLDFIELD CONSOLIDATED

	Per Cent
Old practice:	
(Average of ore Nov., 1909, to Oct., 1910—1.785 oz. gold per ton.)	
Extraction by amalgamation	15.38
By concentration	56.86
By cyanidation	22.03
Total extraction	94.27
By cyanidation from the concentrate	95.44
Total net extraction by amalgamation, concentration, cyanidation and cyanidation of the concentrate	91.68
Recent practice:	
(Average of ore Jan., 1915, to Oct., 1915—0.533 oz. gold per ton.)	
Extraction by concentration	32.28
By cyanidation	58.31
Total extraction	90.59
By cyanidation from concentrate	96.18
Total net extraction by concentration, cyanidation and cyanidation of concentrate	89.36

Example F.—The Belmont mine[41] (Tonopah, Nev.). produces a siliceous silver-gold ore containing the sulphide, sulpho-antimonide and the selenide silver minerals. The ore is crushed in cyanide solution and concentrated prior to cyanide treatment by agitation and filtration. Average assay of ore for a thirty-day period in 1915: Gold, 0.32 oz.; silver, 32.1 oz per ton. Table VII gives extraction results.

TABLE VII. BELMONT MINE

	Silver, Per Cent	Gold, Per Cent
Percentage of extraction by concentration	9.0	6.0
By cyanidation	84.0	90.2
Total	93.0	96.2

Example G.—The San Rafael mine[42] (Pachua, Mexico) produces a siliceous silver-gold ore containing the sulphide and sulpho-antimonide minerals of silver, silver sulphide predominating. The ore is crushed in cyanide solution and concentrated prior to cyanide treatment by agitation and filtration. Average assay of ore over a considerable period of time: Gold, 0.13 oz.; silver, 28.42 oz. per ton. Table VIII gives extraction results.

TABLE VIII. SAN RAFAEL MINE

	Silver Per Cent	Gold
Percentage of extraction by concentration	24.19	(not reported)
By cyanidation	67.24	(not reported)
Total extraction	91.43	97.08

Example H.—The Nipissing mine[43 and 44] (Cobalt, Ont., Canada) produces a low-grade complex siliceous silver ore containing sulphide, antimonide and sulpho-antimonide minerals of silver together with many other impurities, as arsenic tellurium, cobalt, nickel, etc. The ore is crushed in a caustic soda solution followed by extremely fine grinding in the tube mill and reduction with aluminium. After filtration it is cyanided by agitation and filtration. During the early operation of the mill the ore averaged 26.0 oz. silver per ton, but more recently it is of lower grade.

Percentage of extraction by cyanidation	92.0 per cent

Example I.—The Nipissing mine[43] (Cobalt, Ont., Canada) also produces a very high-grade complex silver ore containing upwards of 10 per cent of silver, which occurs chiefly metallic and as dyscrasite with a variable proportion of the sulphide and sulpho-antimonide minerals of silver. The gangue is largely the arsenical minerals of cobalt and nickel containing tellurium, antimony, mercury and metallic bismuth. The ore is ground in a silex-lined tube mill with very strong cyanide solution and a large proportion of mercury. This is followed by cyanide treatment comprising agitation and filtration of the pulp.

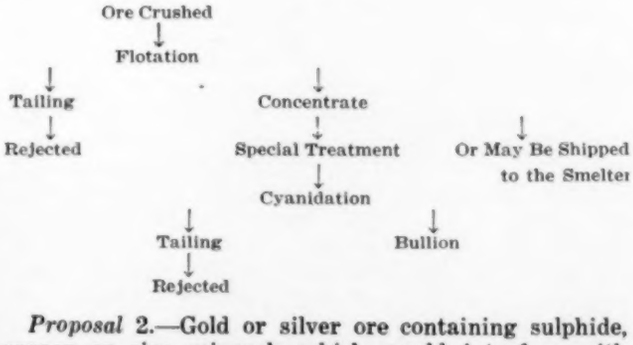
	Per Cent
Percentage of extraction by amalgamation	98.0
Percentage of extraction by cyanidation	1.0
Total	99.0

1. If the ordinary uncorrected assays are used, the bullion recovery frequently approximates 100 per cent.

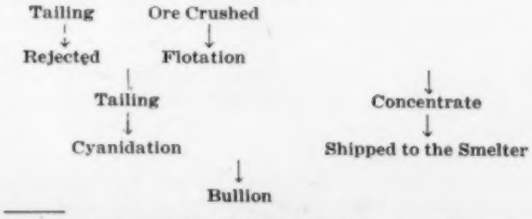
Example J.—The Alaska Gastineau Mining Company[45] (Juneau, Alaska) produces a very low-grade gold ore ($1.25 per ton), having a slate, gabbro and quartz gangue in which occurs free gold, pyrite, pyrrhotite, galena and a little sphalerite. The ore is crushed and concentrated in water by ordinary gravity methods and the concentrate shipped to the smelter. Percentage of the gold extracted 84.0 per cent. This is a good exemplification of the application of the large scale methods which have been so successful in connection with the treatment of the low-grade porphyry copper ores to a gold ore which would be too low grade to be treated by any other method.

The following proposals for certain special cures of ore treatment are intended to be merely suggestive, as they are by no means fully proven nor do they entirely cover the range of possibilities. It is further realized that much investigative work is necessary to properly apply flotation in all cases to the treatment of gold and silver ores.

Proposal 1.—Very low-grade gold or silver ore which, owing to its complex nature and lowgrade, cannot be profitably treated directly by cyanidation:



Proposal 2.—Gold or silver ore containing sulphide, copper or zinc minerals which would interfere with cyanidation:



[41]The Tonopah Plant of the Belmont Milling Co., by A. H. Jones. August, 1915, *Bulletin American Institute of Mining Engineers*, pp. 1731-1758.

[42]The Cyanide Mill of the S. Rafael y Anexas Co., Pachuca, by E. Girault. Transactions of the Mexican Institute of Mining and Metallurgy, Vol. I, pp. 119-128.

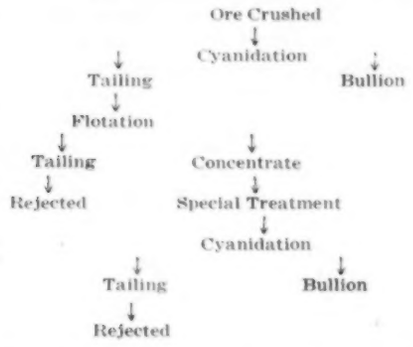
[43]The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, by James Johnson. Transactions American Institute Mining Engineers, Vol. 48, pp. 3-29. Discussion, pp. 30-32.

[44]The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, by G. H. Clevenger. Transactions American Institute Mining Engineers, Vol. 49, pp. 156-179. Discussion, pp. 179-182.

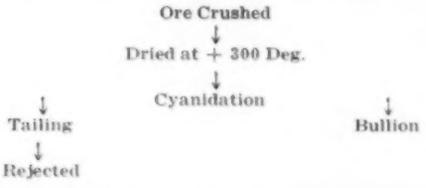
[45]Nipissing High Grade Mill, Cobalt, by R. B. Watson, *Engineering and Mining Journal*, Vol. 94, pp. 1077-1080.

[46]Perseverance Mine and Alaska Gastineau Mill. Robert S. Lewis. Mining and Scientific Press, Vol. III, pp. 397, 400.

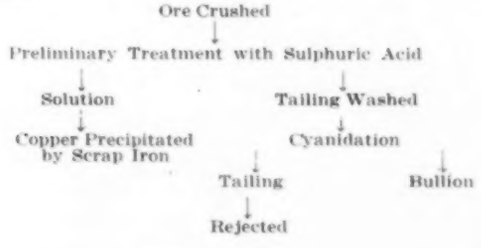
Proposal 3.—Typical gold or silver ore amenable to cyanidation where flotation might be applied to the tailing for the purpose of augmenting extraction:



Proposal 4.—Gold ore of moderate grade containing graphite which interferes with extraction:



Proposal 5.—Gold or silver ore in quartz or other gangue not acted upon by acid, containing oxidized copper minerals in sufficient amount to interfere with cyanidation:



Chemical Exhibition at Urbana Meeting of the American Chemical Society

The chemistry department of the University of Illinois is arranging an exhibition of chemical industries to be held in conjunction with the spring meeting of the American Chemical Society and the dedication of the new chemistry building at the University of Illinois, Urbana, Ill., April 17-21, 1916.

It is the purpose of the committee to emphasize especially the importance of American chemicals and apparatus, and at this critical time to make them better known. A large attendance of chemists is expected at the meeting, in addition to the 2000 students registered for courses in chemistry at the University of Illinois.

No charge is made to exhibitors for floor space; a nominal charge to cover the cost of printing a catalog will be made. Necessary gas, water, electricity and drayage will be furnished free. No admission fee is to be collected at the door.

Prof. H. L. Olin, Chemistry Building, University of Illinois, Urbana, Ill., is the chairman of the exhibits committee.

The Anaconda Copper Co. was forced to close its mines on Jan. 29, owing to the intense cold. The ore was frozen in the bins and cars and it was impossible to transport it to the smelters. After a few days' delay operations were resumed.

The Operation of the Blast Furnace

The Sequences of the Charges which Occur

BY J. E. JOHNSON, JR.

The three articles on the principles of the blast furnace, published in recent issues of this journal, have necessarily to some extent formed a picture of the operation, but we may now consider in more detail the progressive changes which take place within the stack, first, in the raw materials as they descend, and second, in the gas as it ascends.

The Progress of the Raw Materials

While the changes through which these pass shade from one into another so that the number of stages is literally infinite, yet these stages may be divided into four main zones, first, that of drying and warming; second, that of the reduction of the ore, the decarbonating of the stone, and some solution of coke; third, incipient fusion, and the early stages of slag formation; fourth, the reduction of the last remaining portions of the ore from the slag, the completion of the carbonization of the iron, its siliconization, and also its impregnation with the phosphorus and manganese to the extent these may be present.

THE WARMING AND DRYING ZONE

As to the warming and drying zone not very much need be said. In coke furnaces in good practice there is a sufficient amount of residual heat in the gases at their exit from the stock column to give them a temperature of seldom less than 300 or 400 deg., and ranging from this up to a thousand, as we have seen in the article on thermal principles. This provides an ample interval between their temperature and that of the incoming raw materials to provide for the rapid drying and warming of the latter so that the temperature of the charge rises quite rapidly for a short distance below the stock line, and thereafter much more slowly because the temperature interval is much reduced by this rapid warming of the cold incoming stock.

THE ZONE OF REDUCTION

In passing through the second zone, the ore and fuel react with the ascending gases according to the complex laws set forth in the article on chemical principles. In a broad way oxygen is continually taken from the ore and imparted to the gas so that the content of oxygen in the latter constantly rises in its progress through this zone, but in detail we have the complicated reactions whereby the CO of the gas is broken by the catalytic action of the iron ore into CO_2 and C, the latter being deposited as an impalpable dust in the interstices of the charge, thus obstructing the free passage of the gas through it and not infrequently raising the resistance in this zone and the pressure below it. This carbon dust is carried down with the charge to a zone of higher temperature in which the ore can no longer remain in equilibrium with the carbon, so that the latter takes part of the remaining oxygen from the ore, is reconverted into CO, and rises again with the rest of the gas, perhaps to go through the same reaction again.

In this zone also the carbon dioxide of the limestone is driven off by heat, the greater portion of this action probably taking place in the upper portion of the zone. This, of course, results in adding to the gas additional CO_2 which entirely alters the equilibrium relation between the gas as a whole and the ore and carbon of the charge, a portion of the latter being absorbed by the gas to reduce a part of this CO_2 to CO and restore the equilibrium.

Frank C. Roberts several years ago called attention to a case in which a furnace which had been using a coarse, lumpy limestone changed to a rather finely crushed stone, with the result that slips became much more pronounced. The operators finally decided that the change in the size of the limestone was the cause of the slips and upon going back to the larger stone these were accordingly reduced to what they had been before the change was made. Mr. Roberts accounted for this on the hypothesis that with the lumpy stone, because of the time required for the heat to penetrate the large lumps, decarbonization was delayed down to the point where carbon deposition was taking place freely. The result was that the CO_2 from the stone dissolved the carbon dust deposited and prevented it from obstructing the interstices in that zone, leaving a free passage for the gas. With the smaller stone, on the other hand, the CO_2 was driven off at a level above that at which the carbon deposition took place, and the latter was, therefore, not removed by this agency, with the result that a tight zone was formed with slipping as a direct consequence.

I have no experience of the conditions described myself, but the explanation, in addition to being very ingenious, is entirely reasonable and illustrates how difficult it is to determine in advance the effect of any given change on the operation of a furnace, for it is almost universally believed and is probably very generally true that limestone crushed down to a reasonable size is very much to be preferred to coarse, lumpy stone.

Where the ores charged are in the form of lumps, or particularly if the ore be magnetite, the action of reduction is very much slower and the individual pieces of ore may descend well down toward the top of the bosh before they are disintegrated by the reducing action. If the ores are both magnetic and lumpy both causes contribute toward a still further delay in reduction, and we are likely to have solid lumps of ore descending into the bosh and even into the hearth of the furnace, where it is certain that they are reduced by the direct action of carbon with all the loss of both heat and fuel which have been shown to inhere in that action. Such furnaces must inevitably suffer from a fuel consumption materially higher than that of a furnace working on ores inherently more reducible or better prepared. On the other hand, we have come within recent years to realize that ores may cause almost as great fuel loss by being too reducible as by being too irreducible.

With the introduction of large percentages of Mesabi into the burden, the universal experience was that the furnaces worked very much more irregularly with more frequent and much more violent slipping and kicking, with not infrequent explosions of a much more violent nature which we will consider later, and that the fuel consumption was materially increased, even on furnaces working, for the time being, free from slips; so that the increased fuel could not be charged to the irregularity of the descent of the material into the hearth.

The reason for this increase in fuel does not seem to have been understood by anyone until within a year or two. In the summer of 1913 I obtained some authentic data as to the quantity of blast required to burn a pound of coke based on a vast amount of accurate and reliable investigation at one of the largest plants in the country. This amount was 51.4 cu. ft., whereas the amount required to burn a pound of the same coke, if there were no solution loss, under the same circumstances, would have been about 63 cu. ft. Obviously, then, the solution loss under these conditions was about 20 per cent.

I had recently been making some investigations based on the practice described in the paper of Mr. Arnold K. Reese before mentioned, in which the solution loss based on both engine displacements and gas analysis seemed to have been very small, while the fuel consumption for this run which was made altogether on Old-Range ores was, if not a record, at least within a few pounds of any authentic record made until very recent months. What could cause this enormous difference in the solution loss? It may be remarked that making allowance for the solution loss in modern practice the amount of fuel per pound of iron actually burned in front of the tuyeres was almost identical in the two cases, perhaps about a hundred pounds higher in the modern practice, corresponding to a slight increase in the slag volume as compared with Reese's comparatively rich mixture, the difference accounted for by solution alone amounting to about 400 lb. per ton of iron.

The difference in the physical condition of the ore was the only notable difference in the two cases. In Reese's practice the ores were Old-Range, so either lumpy or plastic, and contained no fine sandy material, whereas in the case taken from modern practice the burden was some 80 per cent Mesabi. When a layer of lumpy or plastic ore is charged upon a layer of coke it has little opportunity even after thorough drying out to run down into the interstices of the coke; the lumps, of course, cannot so run and the plastic material when dried out becomes not powdery but lumpy and is, therefore, almost equally unable to penetrate through the coke charge.

With fine, sandy ores the conditions are totally different. As soon as the ore becomes thoroughly dried a large portion of it runs almost like a liquid, especially when in a state of constant agitation caused by the gas current passing up through it, and this portion of the ore rests not only upon the top surface of the coke charge, but runs all through it and makes contact with the coke practically throughout the mass of the coke charge. It must be remembered that ferric oxide and carbon in contact at any temperature above a few hundred degrees are essentially in unstable equilibrium, and that direct action takes place between them, a part of the oxygen of the ore attacking the carbon and carrying it off as CO.

With the Old-Range ores this action could only take place along a single plane of contact, but with the fine, sandy ore scattered throughout the mass the number of points of contact is almost infinitely increased, and obviously the solution loss must necessarily be very much higher under these circumstances than with lumpy or plastic ores.

This, then, taken with the effect of solution loss set forth in the article on thermal principles, forms the explanation of the greatly increased fuel consumption of furnaces working fine ores over those using ores of similar analysis but different physical characteristics.

There has been for many decades, and probably many centuries, a strong prejudice among furnacemen against the use of fine ores, and it is probable that this action just described is responsible for a great portion of this prejudice. Moreover, with certain kinds of fine ore the conditions may be even worse. Furnaces working on fine magnetic concentrates have had the greatest difficulty in securing regular work, irrespective of the care with which the furnace might be burdened and filled, for the reason that the fine ore, as the furnacemen expressively de-

scribe it, "runs ahead." That is to say, under the agitation of the gas current it runs amost like a liquid, not only down through one fuel charge but perhaps through several if the conditions happen to be just right, and owing to its irreducibility, and to its having reached the hearth perhaps several hours ahead of the fuel which should have accompanied it, the furnace is unable to carry the additional load and serious trouble results.

In recent years much progress has been made in treating fine ores before charging them so as to convert them into some form of lumps. This was originally undertaken with the purpose of preventing the very fine portions from being carried out by the top gases and lost, but it has been found that in some cases a real benefit to the operation of the furnace has been brought about, and the reason for this improvement is believed to be that set forth above, that the physical structure of the ore is so changed as to prevent its running through the charge. Perhaps in some cases also, the lumps act as a blanket on top of the coke to prevent fine portions of the ore charge from finding their way down into the coke. Several of these processes reduce the ore partly to the condition of magnetite and thereby decrease its reducibility, but leave the product so porous that the gas and heat have free access to its interior portions, and so prevent its acting as do the large irreducible lumps of magnetite above described.

Throughout this zone of preliminary reduction a continual warming up of the ore takes place, partly by the physical absorption of the heat of the ascending gas current, and partly by the action of the oxygen of the ore in oxidizing the CO formed in the hearth to CO_2, a reaction which gives off more heat per unit of oxygen than is absorbed by the removal of the same amount from the ore and therefore produces a net increase in the amount of heat present.

It will be noted from the equilibrium diagram in the chapter on chemical principles that the equilibrium line for both Fe_2O_3 and FeO descends toward the right or high-temperature side of the diagram, which means that as the quantity of oxygen remaining in the ore is reduced a higher heat and higher percentage of CO in the gas is required to remove the remaining portions, and as the ore descends these are precisely the conditions into which it passes, a continually increasing temperature, and increasing concentration of CO in the gas until at the top of the bosh practically all the carbon in the gas is in that condition.

It is probable that it is because of the long slow slope of these equilibrium lines that this second zone of the furnace requires to be the largest of any in point of size, almost as large as all the other three put together, so that time and space may be allowed for the relatively small but approximately constant reaction pressure (to coin a crude term), to do thoroughly and well its work of preparing the ore as completely as possible for the final operations which take place in the hearth and bosh.

Many other reactions and changes probably go on within this region but from the point of view of operation these are the only ones with which we need to concern ourselves.

THE ZONE OF PRELIMINARY SLAG FORMATION

This zone is one located approximately at the top of the bosh where, as we have previously seen, the materials pass through a pasty stage and finally into a fluid slag, but one by no means of the composition which subsequently is tapped from the furnace.

In a paper of Mr. F. E. Bachman before the American Iron and Steel Institute in October, 1914, occurs the following paragraph which describes in a few words better than anything I have ever seen the action which I believe takes place in this zone.

"My conception of the formation of a blast furnace slag is that at a point approximately 25 feet above the tuyeres the minerals which have the lowest melting point in the ore begin to melt, then the free SiO_2 combines with FeO and with the melted minerals forms a fusible slag somewhat similar to heating furnace cinder. This slag flows down over the pieces of lime and coke, and when it reaches the hotter section of the furnace, the FeO is gradually reduced to Fe and replaced by CaO or by CaO and MgO from the lumps of burned stone. In front of and immediately above the tuyeres the coke ash is set free and combined with the slag produced above, and a portion of the lime not completely fluxed often passes below the tuyeres and is absorbed in the bath of slag in the hearth."

It may be doubted whether in practice other than that described by Mr. Bachman the initial fusion of the materials takes place as high as twenty-five feet above the tuyeres. It seems likely that in furnaces on Lake ores this height is more likely 15 to 20 ft., but leaving aside this detail the fact that slags form in a broad way as described by Mr. Bachman cannot be too strongly emphasized. The furnace does not make initially a clean iron-free slag of the final composition which we tap out, but following the law of most metallurgical operations it makes first of all a slag to suit itself, that is the most fusible one which can be produced from the ingredients present.

Oxide of iron lowers the melting point of lime-silica slags by several hundred degrees, and these are, as Mr. Bachman states, of the general nature of mill cinder, although probably considerably limier than that material. We never see anything of this intermediate slag when the furnace is in normal operation, but when it is in trouble and without sufficient heat in the hearth to furnish the reducing action we obtain from it to our sorrow slags of various and sundry descriptions, some of them thin, flowing almost like water at temperatures literally hundreds of degrees below the running temperature even of charcoal slags, and yet with so little reserve of heat above their melting point that the liquid surface can hardly be seen for the scull which freezes over it instantly when it strikes the air, and a hand-ladle dipped into such a slag for a sample will come out with a heavy crust instantly frozen on its surface, not tough or pasty, but sharp and brittle almost like ice, and as black as pitch itself.

These slags have compositions varying over a very wide range, but the most fusible one that I have ever seen contained a large percentage of iron, something like 35% FeO. In normal operation probably there is not in the zone of initial fusion sufficient unreduced iron to form a slag of this composition, but there is enough to form one of the stiffer and more plastic and more infusible slags such as we sometimes obtain from a furnace when it is limed up, that is, when the fusion point of the slag has been raised so high by excessive lime that there is not enough heat above its running temperature to give it complete fusibility or to remove all the iron from it, it is probably such a slag as this which first forms in the upper portion of the pasty zone. All the observations which we have of the slag from furnaces in normal operation, but taken from above the tuyeres, are of this general character, tough and black, proving the presence in them of some iron still unreduced. This, for instance, is the character of the slags which drip down from above when we are changing tuyeres and coolers or the like.

It is well to note how completely this physical evidence of unreduced iron in the hearth of the furnace agrees with the deduction in the article on thermal principles that the final reduction of the iron is not accomplished until it descends into the bosh. We have here a conclusion reached upon theoretical grounds, born out by all the objective evidence available.

In considering the matter of slag formation there are several points of much importance to be borne in mind. The first two are those mentioned in the article on chemical principles. First, that the formation temperature of slags is generally higher than their fusion temperature which is almost a corollary of the fact that the fusion temperature of the ingredients is far higher than that of the slags. Second, that the fineness of subdivision and intimacy of mixture of the different ingredients has a pronounced effect upon the formation temperature; as it is almost obvious that it should in view of the first condition.

In addition to these it must be noted that in coke practice a very large proportion of the total silica to be fluxed comes from the coke. In most Lake ore practice this probably amounts to about one-half, the other half coming from the gangue of the ore. Now no matter how intimately the flux and the ore may be mixed the lime must be in great excess at the top of the bosh because approximately one-half of the silica contained in the coke is locked up securely therein and is not all exposed to the solvent action of the slag until the coke has all been burned away which as we know does not take place until it reaches the tuyere zone. It is obvious, therefore, that we must have a much limier and therefore more infusible slag at the top of the bosh where the temperature is relatively low than we have in front of the tuyeres where the temperature is at its maximum. Considering the extremely stiff and pasty nature of these excessively limey slags, judging by the occasional ones which we get from the furnace, it is not a matter of surprise that the zone of incipient fusion should be a pasty zone, but rather that it is not a much pastier one, as in fact it probably would be were it not for the presence of the considerable percentages of iron oxide with its strong action in lowering the melting point.

With those classes of ores which contain an excess of lime such as the hard red ores of Alabama we have a somewhat different condition in respect to slag formation than that which exists with ores containing roughly the same amount of silica per unit of iron, but which require to be fluxed by the use of limestone additions. In the first case the lime and silica of the ore are in very intimate contact, and therefore the formation of the slag is greatly promoted, whereas when the silica is in intimate association with the iron alone and must be fluxed by separate lumps of lime derived from the limestone, the tendency is for the silica to unite with the iron in its partly reduced condition and carry it out of the furnace as a black slag, and either more time or more heat must be provided to obtain equally complete reduction in the latter case. This probably accounts for the fact that the soft red ores of Alabama, having virtually the same ratio of silica to iron as the hard ores, are much less desirable in the furnace. Either slower driving or more fuel is required in exact accordance with the above reasoning, for which I am indebted to Mr. W. L. Kluttz.

IMPREGNATION OF THE FUEL WITH SLAG

In connection with the necessity for reasonable fluidity on the part of the slag it may be as well to call attention here to a curious fact of practice which, if it has been observed, has at least not been the subject of any published comment of which I am aware.

Several years ago seeing a pile of carefully screened slag which had been lying in the weather for some time I noticed scattered through it several pieces of material which looked like coke except that instead of being black they were rust-colored, and when picked up were found to be very heavy, perhaps twice the weight of coke. The question as to what this material might be was discussed between the manager of the plant and myself with much interest. Finally we broke open one of the lumps and found that it was coke impregnated with some other material. Mr. B. G. Klugh was chemist of the plant at the time and on being called in he ground up and polished a broken face of one of these specimens; examination immediately showed that a great percentage of the cell spaces, even in the very center of the original lumps, was impregnated with slag. Subsequent investigation at other plants showed that all the coke which came from the tapping hole and had therefore passed through the hearth, was in this condition. In some cases particles of iron as well as slag were found in the center of the pieces of coke. It was noticeable that this slag was always milk-white, irrespective of the slag which the furnace might be making at the time.

When I first began to operate a charcoal furnace I noticed on the casting cinder little piles of coarse fibers about like white horsehair, the individual fibers two or three inches long, and not lying criscross but parallel as if they had come done up in a neat package. I asked what caused this but obtained no satisfaction. A short time after, however, coming past soon after cast, when the cinder was still hot, I noticed pieces of charcoal with a white appearance around their ends in various stages of very slow combustion, no flame, merely a quiet dull red glow hardly visible, by which the carbon was slowly passing away. Closer examination showed that the white appearance at the ends was due to the projection of fibers projecting from the cells, and presently it became evident that the neat little packages of "horsehair" were fibers of slag cast absolutely perfect into the cells of the coal in the hearth, and exposed by the slow burning away of the charcoal as it lay on the hot slag in the open air. Analysis of this material proved it to have the same composition as the main body of the slag made at the same time.

The extreme whiteness of the slag in both these cases is an index of the completeness with which the iron oxide is reduced and removed from it, and suggests the possibility that this extraordinarily intimate contact between the slag and the fuel may be a factor of some practical importance in the final completion of the smelting operation, and certainly it is a factor which can not exist except with a reasonably fluid condition of the slag.

THE ZONE OF FINAL OPERATIONS

This is the region of highest temperature in the furnace extending from a short distance above, to just below, the tuyeres. In it the slag attains its final composition, the coke having all burned away and released the slag-forming ingredients which it contains, while the intense heat suffices, in a proper-working furnace, to deoxidize the last remaining portions of the iron and causes them to enter the iron bath leaving a slag which in good practice contains only two or three tenths of 1 per cent of iron.

In coke practice, owing to the fact that the slags used are on the lime side of the most fusible ratio of lime to silica, additions of silicia cause the production of a more fusible slag, hence the release of the silica from the coke ash, and the increase in temperature, owing to the conditions in the region, both contribute to make the slag more fluid. As we have many times re-

iterated, the critical temperature of the furnace not only in the theoretical sense of that word but in the actual sense that the operation of the furnace depends upon it, is that temperature at which the slag becomes sufficiently fluid to free the remaining traces of iron, both shot and oxide, which it contains, and sufficiently mobile to cover the coke and carry away practically all the sulphur it contains, as fast as the latter is disengaged, thus keeping all but small traces out of the iron.

We see, therefore, that the slag has only to reach this condition in its final stage, but it is absolutely essential that it be reached, if not, evil consequences will follow very rapidly. First, the sulphur is no longer sufficiently absorbed by the slag, and deleterious quantities enter the iron lowering its value and finally reducing it to a non-merchantable commodity. Simultaneously with this begin considerable losses of iron carried off with the slag, sometimes as shot, more often in the form of ferrous oxide dissolved in it, and finally as an inextricable mixture of bad iron and bad slag which do not separate instantly with a clean line of parting like oil and water, but which, partly for physical and partly for chemical reasons, remain mixed together if not dissolved one in the other.

Following these conditions, the slag becomes too stiff to be flushed from the furnace through the ordinary opening of one to two inches in diameter, and the water-cooled monkey through which this operation is performed must be removed bodily making a hole some 6 to 10 in. in diameter through which the slag will flow unless the furnace is lime-set or in serious trouble from lack of heat. "Lime-set" was a term used principally by the furnacemen of an older generation, before the simple principles of furnace burdening were as well understood as they are now, to describe the condition to which furnaces were not infrequently brought by the addition of more and more lime to accomplish certain results in the kind of iron produced, or through a misunderstanding of some abnormal condition of the furnace and attempting to correct it with what turned out to be an overdose of lime.

The fusion temperature of slags with increasing lime content goes up with great rapidity, and after a certain point is reached the free-running temperature of the slag becomes higher than any temperature obtainable within the hearth of the ordinary blast furnace, with the result that the slag cannot be removed, and if the furnace continues to be blown this infusible slag rapidly builds up in the hearth and bosh making it more and more difficult for the blast to penetrate from the tuyeres up through the stock column until finally the hearth and bosh of the furnace are almost filled solid with this excessively limey material, which when it is exposed to air or cooled off, not infrequently breaks down much like slacked lime.

In charcoal furnaces occasionally we reach a similar condition from a cause exactly the opposite since slags containing a great excess of silica while they are not as infusible as those containing a similar excess of lime, are extremely stringy and viscous, with the result that at the comparatively low temperatures of the charcoal furnace they may build up on its walls almost as badly as do excessively limey slags in the coke furnace. But the temperature range through which they may be made to flow, even if slowly, is wider than in the case of the limey slags, and charcoal furnaces are therefore not so liable to become completely "bunged up" in the expressive language of the furnace man, from this cause as were coke furnaces a few years ago.

In this zone important changes occur in the iron itself. We have already seen that the silicon and phosphorus can only be reduced in the presence of incandescent carbon at the highest temperatures obtainable in the furnace, and when the iron is present to absorb them. The same thing is true of the manganese in a less degree. That is to say, a furnace charged with an ore of phosphorus or silicon without iron would not reduce any of those materials, and a blast furnace operated in the ordinary way charged with manganese ore without iron would reduce little or none of that element, although by material changes from regular iron furnace practice it can be made to do so. In order then that these elements may be reduced, it is necessary that the iron be present as such so that its affinity for them may assist the action of the incandescent carbon and the high temperature in reducing them from their oxides.

These actions, therefore, go on exclusively in the hearth. It is probable that a large portion of the carbonization in the iron also goes on at this point because it is quite difficult, by any other process, to make iron absorb as much carbon as the product of the blast furnace ordinarily contains. Even the electric furnace with its enormous temperature has found some difficulty in securing the conditions which would give the carbonization of iron necessary to constitute real cast iron, while other methods of melting have generally not been able to raise the carbon much, if any, above 3 per cent.

In the hearth of the furnace we have the condition that iron is subdivided into extremely small streams, trickling down over incandescent coke, in an atmosphere as reducing as it is possible to produce, and at the highest temperature of the furnace. Under these conditions the completion of the carbon absorption takes place.

As to how much carbon is absorbed in the regions above this we do not know. We do know that iron exposed to intimate contact with fine carbon over long periods of time and fairly high temperatures, absorbs 1 or 2 per cent of carbon, and when the ore is of a nature to produce the deposition of carbon dust from the gas upon itself, we have the conditions under which carbonization may easily take place, but so far as I know no determinations have ever been made as to the extent of this action.

In addition to these, other actions go on in the hearth, and some of them we find it extremely difficult to account for. One of the most notable problems is the composition of the salamander taken from the hearth when the furnace is blown out. Salamander is a term covering literally a multitude of sins.

The salamander of the old-fashioned charcoal furnace consisted of what geologists would call "pudding stone," of charcoal slag of all colors and compositions, interlaced and tied together with a stringy or sponge-like mass of iron.

In coke furnaces we find masses not unlike this in general characteristics except that imbedded coke is rare, and the iron portion of the structure is less sponge-like. The intimate mixture in this case is of iron, slag, graphite, and sometimes firebrick, but in the center of the coke furnace we usually find a depression well below the level of the original brick hearth of the furnace filled with iron, which, below the surface, is quite unmixed with slag. The body of solid iron varies much in size and is generally very graphitic around the edges, but frequently extremely dense and solid in the center. Sometimes this mass consists only of the frozen pool of metal last formed in the furnace which could not be removed from it because of its depth and of the absence of a sufficient blast pressure to lift it out through the tapping

hole. But usually in addition to this there is another mass below it, and generally quite distinct from it, which likewise consists of solid iron, but whose composition bears apparently no relation whatever to the composition of the iron made during any portion of the blast. For instance, it is well known that the affinity of iron for phosphorus is so great that practically all the phosphorus introduced into the charge is taken into the iron, yet these salamanders often contain only a small fraction of the phosphorus contained by the iron made during the blast. The manganese is generally low, the silicon also is quite low, and the carbon which in cast iron runs practically always from 3½ to 4¼ per cent; I have seen in one of these salamanders down to about 1.5 per cent.

This material was really a poor grade of high-carbon steel of which the crystals were distinctly bent when the broken faces were slid over one another, and of which a piece taken to the blacksmith's shop could almost, but not quite, be forged.

I am advised by John M. Reese that he has had a practically identical experience.

Comparing the analysis of such a material with the pig iron from which it has presumably been produced, we can only reach the conclusion that there has gone on here a slow but none the less effective oxidizing action at the bottom of the bath although cut off from access to the blast by a layer of iron or slag, or both, and at cast time when the hearth is relatively bare, covered by a mass of coke several feet in depth below the tuyeres, conditions apparently precluding any possibility of access to this region by the oxygen of the blast.

The cause of this peculiar action is not fully understood as far as known to me, but it is impossible to resist the conclusion that the oxidation must be carried on by slag not completely reduced; of the kind which we always find trickling down the walls whenever we have occasion to investigate conditions.

It would seem necessary to suppose that oxygen in this form is carried down into the hearth, that it is cut off from the reducing action of the coke column by the fact that the latter is in part floated on the surface of the charge as described in a previous article, and that this slag which is known to be a very active oxidizing agent satisfies its desire for oxygen at the expense of the metaloids in the plastic but not quite fluid mass of iron in the bottom of the furnace.

The mechanism by which this action is carried out I do not pretend to understand. If anyone will work it out and publish the results he will satisfy the acute scientific curiosity of a great number of his professional brethren by so doing.

The Asphalt Primer and Colloidal Catechism is a little pamphlet issued by the Barber Asphalt Paving Company of Philadelphia. It is an exceedingly clever piece of propaganda, interestingly written and catchingly illustrated. It starts with ancient road construction, going back to King Cheops and the Via Appia of the Romans, compares ancient and modern road construction, and deals with asphalt and its function in a paving mixture. The fact that Trinidad Lake asphalt contains colloidal matter is emphasized and simple explanations are given of such terms as colloids, surface energy, and adsorption; and what they have to do with paving mixtures. In a most skilful manner the non-suspecting lay reader is led to absorb some highly modern chemistry and the primer winds up with a tribute to colloidal chemistry for placing the formulation of paving mixtures on a scientific basis and for accounting for the established excellence of Trinidad lake asphalt.

Liquid Chlorine*

BY DR. G. ORNSTEIN

All are familiar with the element chlorine—its history, manufacture and characteristics—so that it will be hardly necessary to go into details on this subject.

CHEMICAL PROCESSES FOR PRODUCING CHLORINE

The first process employed commercially was the same which we have all used as students in the laboratory—namely, that of generating it from manganese and acid, which may be either muriatic or a mixture of salt and sulphuric acid.

This process was made eminently practical later on by Weldon's invention of recovering the manganese by means of addition of lime to the waste liquor, blowing air through the mixture, and returning it in this regenerated form to the process. This Weldon process is the only purely chemical process operated on a commercial scale, which has furnished a chlorine gas of sufficient concentration to be used for liquefaction. The Deacon process in which hydrochloric acid fumes are decomposed in the presence of air and a contact substance, such as copper chlorides, yields a weak gas which is not practicable for the purpose of liquefaction—at least not directly.

To overcome this drawback in this process and in other instances where a weak chlorine gas is obtained, several processes protected by patents have been worked out, particularly by Goldschmidt and Sperry, which aim at absorbing the chlorine from a weak mixture, the former by stannic chloride, the latter in the form of chlorine hydrate, thus separating it from the diluting impurities and later on decomposing it again by application of heat for the purpose of producing pure chlorine. In the Acker process, in which an extremely poor gas was obtained, one went even so far as to first convert the chlorine into bleaching powder and subsequently decompose it by acid.

However, it seems that none of the chemical processes are operated any longer for the purpose of liquefaction, except perhaps in a few individual instances, and the chlorine liquefied nowadays, in this country as well as abroad, is almost entirely produced by electrolytic processes.

ELECTROCHEMICAL PROCESSES FOR PRODUCING CHLORINE

The electrolytic decomposition of brine or muriate of potash is well known in principle. It consists in passing an electric current of high amperage through an electrolytic cell, in which the anode is made of carbon, or what is much to be preferred, Acheson graphite, or fused iron oxide, while the cathode consists of iron, either in the form of a perforated sheet or wire screen, or in some instances a solid sheet depending on the details of construction.

When the electric current is passed through the cell the brine is split up into its components; chlorine will escape at the anode and caustic soda or caustic potash will be produced at the cathode. The products of electrolysis have to be kept apart, as otherwise they will combine, forming hypochlorite and eventually chlorates, and quite a number of constructions have been devised and patented to prevent this re-combination. In the larger number of constructions diaphragms of various designs are employed—for instance, in the Gibbs, Townsend, McDonald, and Billiter cells the basis of the diaphragm being mostly asbestos. In others they are kept apart by gravity and by causing a constant flow

*A paper presented at the meeting of the American Electrochemical Society (N. Y. Sec.), on February 11, 1916 (in joint session with the New York Sections of the American Chemical Society and the Society of Chemical Industry).

of liquid in the direction from the anode to the cathode. This principle is employed, among others, in the "bell process."

A third principle is employed in the mercury cell, in which the cathode is formed by mercury instead of iron, absorbing the sodium or potassium under the influence of the electric current, and this in turn is released again by water outside of the electrolytic cell in the form of caustic. The pure mercury is then again returned to the cell and reloaded at the cathode—but no matter of what construction the cell may be, in all instances a highly concentrated chlorine can be obtained which, with graphite or iron oxide anodes being less subject to electrolytic oxidization than carbon, should easily yield gas of 97 to 98 per cent purity, the balance being carbon dioxide, electrolytic oxygen and a slight trace of air. With iron oxide electrodes, of course, no carbon dioxide can be formed, but the amount of electrolytic oxygen will be correspondingly higher. However, only in a few cells a gas of this purity is actually obtained, on account of mechanical defects, such as leaks in the cell covers or pipe lines, or negligence of the cell attendants.

LIQUEFYING CHLORINE GAS

For the liquefaction of chlorine it is of great importance that attention should be paid to generating a highly concentrated gas, because otherwise the percentage which can be liquefied from a certain volume of chlorine will decrease rapidly with each per cent of impurities.*

The balance, which still contains between 55 and 65 per cent of chlorine, is called blow-off gas and is released from the liquefying machinery, from time to time, when the pressure exceeds certain limits. It is generally conveyed to the bleach chambers; this is the preferred way of both using and getting rid of it, as on account of its dilution it can be used only for a limited number of other purposes.

In any case absorption capacity of some kind has to be provided for, and it therefore appears that a chlorine liquefaction plant can be operated to advantage only in connection with the manufacture of other chlorine products.

HISTORY OF LIQUID CHLORINE

A great many attempts have been made to liquefy chlorine ever since the beginning of the nineteenth century, but until recently they were largely confined to laboratory experiments without any practical value.

The first liquid chlorine was obtained in 1805 by Northmore, and as early as 1823 Faraday demonstrated the well-known experiment which is found in every text book, namely, the decomposition of chlorine hydrate in a sealed and bent glass tube, the hydrate being heated in one branch of the tube and the escaping chlorine being liquefied in the other branch, which is cooled in a freezing mixture. But until after 1880 nothing had been done which could justify any hope of success for working out a commercial process.

Between 1880 and 1890 a number of processes were worked out which at least made an attempt to produce liquid chlorine on a commercial scale. For instance, Vautin attempted to compress chlorine gas by pressing air into tanks containing chlorine and at the same time cooling it down to a low temperature.

Others, for instance Hannay and Marx, based their processes on the decomposition of chlorine hydrate in a closed vessel. Heinzerling attempted to reach his object by first compressing chlorine, cooling it to a low temperature and then liquefying it by sudden expansion. But none of these attempts were successful and commercial liquid chlorine remained an unrealized dream until the arrival of the process of Knietsch, of the Badische Company, who in 1888 placed the first liquid chlorine on the market. He was the first to realize that it was of fundamental importance for the commercial liquefaction of chlorine that chlorine be used which is at once highly concentrated and free from moisture—two points which the former inventors had overlooked. He also found that chlorine carefully dried did neither attack iron nor a number of other metals employed for building commercial machinery, and through an ingenious device of a so-called liquid piston he was able to put the chlorine under a high pressure without any danger of leaking through the stuffing box around the piston, causing corrosion, etc., For the liquid media he used a column of petroleum around the piston, and below this and in the other branch of the U-shaped cylinder he used concentrated sulphuric acid. The liquid piston was lifted up and down by a plunger moving in the petroleum and thus sucked in and compressed chlorine gas which through a separator was carried on to a condenser where, under the influence of low temperature, it liquefied and was then filled in to the shipping containers.

He also heated the sulphuric acid by a steam or hot-water jacket in order to reduce the absorptive power of the acid for chlorine gas.

According to my knowledge most processes which are now in operation are still based on these fundamental principles. That is, all of them are using a liquid for compression, and while they may have been simplified in some way, for instance by omitting the petroleum and not heating the sulphuric acid or by making changes in other directions, compressing the chlorine by entrainment, or in two or three-stage compression, yet in almost every case the principle has been maintained.

There are several processes on the market, for instance that of the Linde Company of Germany, by which liquefaction is effected simply by cooling to —45 deg. C. or below, without any pressure. But the drawback of this process is that the percentage liquefied is lower throughout than that obtained with the combined system of compressing and cooling.

Regarding the relative merits of the various processes I must hesitate to express an opinion. There is so much secrecy observed in this field, as in all lines of chemical industry, that I do not know enough about the details of construction and operation of most of the other manufacturers to warrant my passing comparative criticism on this subject.

In one point I feel "the people who know" will agree with me—stick to what you have got and have tried out in the course of years and of which you have studied all the little knacks and moods and—never mind what excellent results the other fellow "says" he gets. I would rather hunt for trouble in a twelve-cylinder automobile motor than tackle a chlorine compressor when it is in the mood of the "morning after the night before."

The liquid chlorine is shipped in steel cylinders containing about 100 lb. net and of a tare weight of 70 to 100 lb. In large quantities it is also shipped in tank cars. The pressure in these containers varies with the temperature from about 70 to 100 lb. per square inch at ordinary temperatures. But the pressure will drop rapidly on account of the heat consumed by evaporation if chlorine is drawn off continuously even in relatively small quantities.

In one place where chlorine was drawn off continuously for the purpose of water sterilization at a room

*The author showed curves illustrating the influence of concentrations as well as of pressure and temperature.

temperature of about 2 deg. C. the pressure dropped in 24 hours from around 55 to 15 lb., although not more than ¾ lb. per hour were taken from a set of four cylinders, or 3 oz. per hour per cylinder. This cooling or freezing action can be easily counteracted by playing steam on the cylinder until the frost which will form has disappeared, or by placing it in a tub with warm water or in a cabinet which is kept warm by artificial heat or similar means.

LIQUID CHLORINE IN THE UNITED STATES

In this country commercial liquid chlorine was unknown until the year 1909. There had been imported prior to this date quantities of liquid chlorine from Germany, but these were used solely for experimental purposes on a more or less extensive scale, but the United States did not begin any manufacture until 1909.

It was in the beginning of this year that the Electro Bleaching Gas Company, after about two years of experimenting, erected the first compressors, and a short while later two more chlorine concerns started in operation, the Goldschmidt Detinning Company and the Castner Electrolytic Alkali Company. The former one used the product largely for its own purposes, namely, the detinning of tin scrap, whereas we had to find and build up a market for this entirely new product.

The two most important fields to which the liquid chlorine was to be introduced were the bleaching of vegetable fiber, such as cotton, flax and paper pulp, and chlorination of all kinds, such as ores, organic compounds, pharmaceutical products, etc.

Above all, we found it necessary to instruct our purchasers how to use the liquid chlorine for their various purposes, and we still recall with considerable pleasure the many comical occurrences which happened at that time. In one textile mill our friends apparently thought the cylinder contained something like a liquid bleaching powder which they had been accustomed to use for generations. So they simply attached a hose to the valve outlet and tried to empty the cylinder into a vat containing water for diluting to proper strength and then they opened the valve. The result may be easily imagined. After a minute or two there was nobody left in the mill who could be induced to enter the place and shut off the valve. The letter we received was all but complimentary. Many other episodes of a similar nature occurred.

Therefore, before we could market the product we had to instruct our purchasers how to use it, and we were compelled to make demonstrations of how to prepare the bleach liquor and how to bleach yarn with it in one mill after the other. Finally we succeeded in convincing a number of the largest and most conservative mills of the decided advantages of liquid chlorine as compared with bleaching powder, and these mills for the past four or five years have definitely abandoned the bleaching powder in favor of the liquid chlorine.

BLEACHING SOLUTIONS MADE FROM LIQUID CHLORINE AND BLEACHING POWDER

At the beginning we had to convince many chemists that the amount of available chlorine present in a bleaching solution prepared with liquid chlorine and alkali, for instance soda ash, was equal to the full amount of chlorine passed in. They were basing their objections on the formula:

$$Na_2CO_3 + Cl_2 + H_2O = NaCl + HOCl + NaHCO_3$$

or if caustic soda was used,

$$2NaOH + Cl_2 = NaOCl + NaCl;$$

they insisted that these equations proved that one-half of the chlorine was converted into NaCl and thus wasted for bleaching purposes.

The assumption was that in bleaching powder the available chlorine was present in the form of a $CaOCl_2$. As a matter of fact this is the most accepted formula for the powder, so that the impression is given as if all the chlorine present were in the available form; but upon dissolving this will be split up according to the equation,

$$2CaOCl_2 = CaCl_2 + Ca(OCl)_2,$$

so that one-half of the chlorine is converted into the chloride and the titration with As_2O_3 proceeds according to the formula

$$Ca(OCl)_2 + As_2O_3 = As_2O_5 + CaCl_2.$$

Thus it can be seen that the so-called available chlorine does not represent the actual amount of chlorine present in the hypochlorite, but only one-half of it, and is rather an expression for the available *oxygen* in the same compound which if expressed in chlorine value must, on account of the bivalency of the oxygen and the monovalency of the chlorine, naturally have twice the chlorine value.

It now follows from comparison of the above equations that the chemical reactions of bleaching powder solutions and solutions made with liquid chlorine and alkali are identical, and the fact can no longer be astonishing that if liquid chlorine is used for the manufacture of bleaching solution, an available chlorine content is obtained substantially equal to the number of pounds of chlorine absorbed by the solution.

We were able to show—and in connection with this wish to acknowledge the valuable assistance rendered us by Dr. J. M. Matthews—that bleach solutions thus prepared gave decidedly better results than bleaching powder. We could demonstrate that only one-half of the amount of available chlorine and even less is required to be present in order to give the full bleaching results which formerly had been obtained with chloride of lime, while the tensile strength of the goods is correspondingly improved.

A large reduction of the so-called sour, namely the use of weak sulphuric or muriatic acid after the bleaching bath for the purpose of destroying the last traces of residual chlorine, represented a further improvement in this important respect. The explanation for the increased efficiency seems to be that in case a bleach is used in which the chlorine has been combined with soda ash the presence of free hypochlorous acid causes a more rapid and more efficient action on the fiber than the neutral calcium hypochlorite. In addition any bleach prepared with soda as a basis and with entire absence of lime salts seems to have a higher penetrating power on the cloth on account of no substances being present which are capable of forming insoluble preciptates on the fiber. This theory is substantiated by the observation that a hard twisted yarn or cloth requires less available chlorine as compared with the necessary quantity of bleaching powder solution than a material of a looser texture.

OTHER ADVANTAGES OF LIQUID CHLORINE

Another large field in which liquid chlorine is now used extensively is for chemical chlorination processes, in which the quantities consumed are not large enough for warranting the concern to build a branch factory near the chlorine-producing concern for receiving the gas directly from the cells, so that the use of liquid chlorine, which is easily transportable, is the only other choice.

A number of manufacturers have tried to make the chlorine themselves from either bleaching powder and acid or manganese and acid, but in most instances they have found that it was much more to their advantage to purchase liquid chlorine ready made. The reason is

not only the cost of the raw materials, which in either case exceeds the price at which liquid chlorine can be purchased, but there is also the advantage of the liquid chlorine being absolutely dry, chemically pure, and under a pressure sufficiently high to be easily conveyed to any point of application and pressed through liquids to be treated without the troublesome medium of blowers, fans, pumps, etc.

The valve of the chlorine cylinder, also, permits of an easy and accurate regulation, whereas the above-mentioned chemical processes when once started cannot be regulated at will unless a gasometer is used in connection with them; this only introduces another source of trouble because as the only sealing medium concentrated sulphuric acid must be employed.

Of the advantages just cited, its purity is one of the most conspicuous characteristics of liquid chlorine. When a fresh cylinder is opened it will usually yield at once a gas which is 96 to 98 per cent pure, and after several pounds, from 3 to 5, have been taken out, it goes up to 99 per cent, and after another pound or two it will be at 99.7 and 99.98 per cent, and will stay there until the last drop of liquid has evaporated from the cylinder. In view of these figures, which can be easily substantiated at any time, liquid chlorine may justly be called one of the purest, if not *the* purest, chemical on the market. The initial slight amounts of impurities represent the small left-over traces of air, etc., which were in the gas coming from the cells.

LIQUID CHLORINE FOR WATER STERILIZATION

A third large field for the use of liquid chlorine has been opened up during the last three years, namely its use for the sterilization of city water supplies and sewage. In this field as well as in the bleaching industry liquid chlorine had to compete with its old competitor, bleaching powder, but we are proud to say that it has made a pretty good showing, even without making allowance for its youth and for the lack of experience.

After some experimental work had been done previously the first commercial machines were installed in the Western New York Water Company plant at Niagara Falls, N. Y., in November, 1912, where we received valuable assistance from F. H. Huy, general manager of the Western New York Water Company's plants, and in Wilmington, Del., in January, 1913, where they are still in continuous satisfactory operation. These machines are based on the principle of absorbing a measured quantity of chlorine in a small amount of water and subsequently adding this solution to the water to be treated. This so-called absorption process has the advantage of an even and rapid distribution of the sterilizing agent throughout the whole body of water. This advantage is particularly apparent in large installations, such as the Torresdale filter plant in Philadelphia, where from 220,000,000 to 330,000,000 gal. of water are filtered daily.

It is furthermore of advantage in plants in which the water to be treated is under high pressure—which in some places exceeds the pressure of the liquid chlorine itself. In these installations the application of the chlorine in the form of a solution which is forced into the main by means of a specially constructed pump is the only form in which the problem can be solved.

In the field of water sterilization the Leavitt-Jackson Engineering Company is deserving of particular mention on account of the original principle involved, which consists of feeding the chlorine into the water by weight by a balanced beam construction acting on the chlorine feed valve. This is an excellent construction mechanically, but not sensitive enough for the minute quantities of chlorine which are required by far the largest number of water works.

The principle used in our first machine comprises the pressure regulation by a reducing valve or by two switched in series, the measuring of the rate of flow by the pressure difference on both sides of a calibrated orifice caused by it, and the subsequent absorption of this measured quantity in water prior to its addition to the main flow of water.

Under the conditions in which the chlorine in the absorption tower is discharged against atmospheric pressure it is unnecessary to connect the U-tube to the down-stream side of the orifice, as would have to be done were the chlorine to be discharged against pressures higher than atmospheric.

We are also using a rate of flow meter for measuring the chlorine.

Through constructions along these lines it has been possible to effect an accurate regulation of chlorine gas to any quantity as small as 1/40 oz. per hour and as high as 25 lb. per hour, irrespective of any variations of the pressure in the cylinder, which in the course of operation may drop from 100 to 15 lb. without influencing the rate of feed.

To realize thoroughly the importance of employing a sterilizing agent which can be added under perfect control as to its quantity, it is sufficient to point out that it is necessary to add it to a continuously flowing stream of water and that it has to be mixed and intermingled with it continuously, thoroughly and evenly. This is not possible with chloride of lime, because the devices used for this purpose, such as orifice boxes, etc., will after a short time be partly or entirely plugged up by a lime deposit and will be in full working order only for a few days at a time. We have also succeeded in developing apparatus capable of automatically varying the amount of chlorine added in proportion to the rate of water flow.

Further advantages which may be cited in favor of liquid chlorine are: absence of taste and odor in limits very much wider than those prevailing for bleaching powder—no sludge or dust—smell or other nuisance, and particularly its *efficiency*, which is two to three and one-half times as high as the same amount of available chlorine present in bleaching powder. In other words, 1 lb. of liquid chlorine has approximately the efficiency of 6 to 10 lb. of chloride of lime.

In these various directions we had first to overcome considerable prejudice on the side of the communities, particularly as regards taste and odor. To what extent prejudice may sometimes run away from common sense may best be learned from the fact that complaints about chlorine taste and odor came pouring into the office of the Board of Health in Buffalo—four weeks *before* our machines were installed.

The results obtained by this process and applied by these machines have been highly satisfactory, as can especially be gathered from the Board of Health records of the various communities which in cities which formerly had an exceedingly bad record for typhoid fever are now reported as being almost or entirely free from it, and in no case, wherever typhoid fever then occurred in such a community, has it been possible to trace it back to the water used, but the reasons were always found in causes other than the city water supplies, such as milk, impure food, etc.

LIQUID CHLORINE AND THE WAR

This evening's papers deal with electrochemical war supplies and I, therefore, have to devote some space to the use of liquid chlorine for war purposes.

The war has naturally had considerable influence on

the liquid chlorine trade, though the causes are entirely different from those which are responsible for the demoralization of prices of other chemicals.

For the latter the present situation is caused simply by the fact that they were hitherto imported from abroad and only small quantities, if any, manufactured here. Now the supplies being entirely shut off, prices, of course, have gone sky-high. Liquid chlorine, on the other hand, is a domestic material which, therefore, could not be influenced by the shutting off of imports as to the quantity available. The increase in the trade and the price of liquid chlorine must principally be traced back to the shortage of bleaching powder, of which formerly more than 50 per cent was imported, and on account of it being shut off its price has risen from about $23 per ton to figures between $200 and $300, whereas the price of liquid chlorine has only increased from $160 or $200 per ton to $240 or $300. Taken in figures of available chlorine, which is the compound by which, apart from other considerations, the comparative values of the two materials are measured, this is now worth in bleach containing 35 per cent available chlorine $570 to $860 per ton, whereas in liquid chlorine, which can be almost quantitively converted into the available form, it can be purchased for no more than the price of the liquid itself.

How long these conditions will continue to prevail is hard to say. Several of the big manufacturers are enlarging their plants and their bleach capacity, which may relieve the present shortage to some extent, but hardly enough to reduce prices to anywhere near normal. The enlargement of plants, however alluring, is carried out very conservatively, realizing that after re-establishment of peace the imports will begin again from Europe, particularly Germany and England, perhaps not immediately, as some assume, but soon after.

In the meantime the price of liquid chlorine may still travel a long way upward before approaching the present value of available chlorine in bleaching powder.

Due to this condition, namely, a price enormously higher for bleach than for liquid chlorine, manufacturers have been much tempted to divert part of the chlorine from the liquefying machinery into the bleach chambers, and a certain stringency may have been caused by this, as well as by the exportation of moderate quantities for war purposes.

From the reports we have received from the other side, liquid chlorine as such has been used as a weapon for incapacitating the enemy. To what extent these reports are strictly true and what part liquid chlorine as such has played in it, we do not know.

I have read only one report in one of the medical journals which seems to be entirely unbiased and therefore in fair accordance with the facts, free from newspaper sensationalism.

The report shows more or less severe affections of the breathing organs, so severe in some instances that it seems incredible, according to my experience, that they should have been caused by inhaling of chlorine gas. Of course, I do not doubt that chlorine can be very detrimental to the human system in a closed room, from which there is no escape, but in the open air with a wind blowing, no matter how gently, it seems next to impossible that any suffering beyond that of temporary coughing spells, vomiting, etc., could occur.

It has been suggested from various sides, and it is undoubtedly a fact, that the chlorine if employed is mixed with other ingredients, such as bromine, sulphur and others, which would cause more severe and more lasting irritations than chlorine alone. I know personally of one instance in which sulphur has been used in connection with liquid chlorine.

We naturally have had a long experience in this line, particularly at the start, when we had to struggle with a good many imperfections, leaks, etc., and never in any instance have we seen any lasting detriment suffered by any of our men.

I have also had several years' experience in Germany along the very same lines, and there also I have never been able to observe one single case of this kind. To appreciate this statement fully you ought to see into what kinds of atmosphere our men had to go sometimes without using any protection whatsoever, and when they had swallowed half of our production and were coughing violently they took a drink of Rexalls' A. B. C. seltzer or bromo seltzer and a few minutes later were back in the plant. These compounds, as well as any other of a stimulating or nerve relaxing influence, seem to be the best antidote for chlorine—among them a good stiff drink of whiskey.

If these are not available, inhaling of alcohol, chloroform or tetrachloride of carbon vapors spread out on a handkerchief will give speedy relief, but with the two latter ones one should be careful so that no narcotic effect may take place.

The inhaling of ammonia, which is recommended in several text books, should be absolutely rejected, because in most instances the irritation grows only worse instead of better.

Prevention is better than a cure, and the best prevention of inhaling too much chlorine is not to be afraid of it, and if a whiff of it gets into the nose, not to stop breathing abruptly. If you do and if you cannot get out of the room before you have to breathe again, you are bound to take a deep breath which will carry the chlorine to all the tender parts of your lungs and a violent coughing spell will be the result.

If, on the other hand, you continue to breathe—lightly, without taking a deep breath—you can stand a pretty strong chlorine atmosphere for minutes at a time, without any muzzle, safety helmet or other protection. Be careful, however, not to inhale the white fumes which are formed by ammonia and chlorine, in the course of trying to locate chlorine leaks with a rag soaked in ammonia. The coughing spells and the effect on the eyes caused by them are very much worse than with chlorine alone and last longer. Also wet chlorine such as emanates from a saturated chlorine water seems to have worse effects than the dry, although only slightly more so.

We have in our plant an oxygen safety helmet, but except for purposes of keeping our men trained in its operation, I have never seen it in actual use.

Another purpose for which undoubtedly quantities of liquid chlorine have been shipped abroad from this country is for the manufacture of picric acid, which, in this instance, can be made by first producing chlorbenzol, nitrating it, and then replacing the chlorine by hydroxyl by treatment with alkali.

These are the two only purposes for which, as far as I know, liquid chlorine has been used for war purposes. However, as you see, they are rather limited in their effects and, therefore, hardly deserving of the prominent place accorded them in the newspapers.

In general, I may say—and I hope I have been able to demonstrate it in this paper—that liquid chlorine is much more important for peaceful than for war purposes, and we are taking considerable pride in the fact that in the past few years we have been able—instead of causing destruction of human lives—to save them from the ravages of typhoid fever and in this way to prove that instead of being an enemy, liquid chlorine is becoming more and more a savior and a boon for mankind.

Recent Chemical and Metallurgical Patents

Gold and Silver

Sodium-Amalgam Precipitant for Cyanide Solutions.—It is known that metallic sodium may be diluted with mercury so that it will replace various metals from their solutions, the metals thus displaced combining with the mercury. This principle is applied by HANS FOERSTERLING and ARTHUR L. HALVORSEN of New York City, in the patented use of sodium-amalgam for precipitating gold, silver and copper from cyanide solutions. The apparatus designed for applying the invention is shown in Fig. 1. It consists of cylindrical casing 1, of soft steel, an axis 2 bearing an inclosed drum 7 on which, at suitable intervals, are annular soft steel plates 8 having perforations 8a for the passage of solution. In the

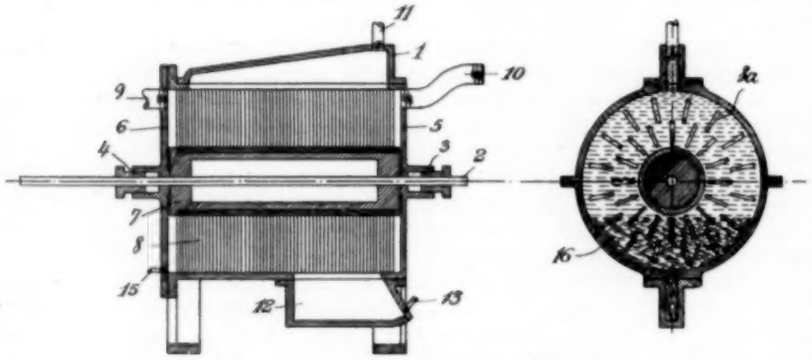


FIG. 1—SODIUM AMALGAM PRECIPITATING APPARATUS

upper part of the casings are pipes 9 and 10 for the inflow and outflow of solution, while pipe 11 serves as an exit for gases. At the bottom is a pocket 12 into which material can be introduced through pipe 13. The axis, drum and attached plates are adapted to revolve through a bath of sodium amalgam in the lower part of the casing, and carry thin films of amalgam upward through the solution flowing through the casing. As the metals are precipitated, cyanide is regenerated. The precious-metal amalgam can then be drawn off through pipe 15, while new supplies of sodium and mercury are added through 13.

Application to the precipitation of silver-bearing solution is illustrated thus: The solution contains 8.5 oz. silver and 0.095 oz. gold, 3.7 lb. free NaCN and 0.5 lb. $Ca(OH)_2$ per ton. As it passes through the apparatus at the rate of 100 tons per day, it comes in contact with the thin films of sodium amalgam on the plates and silver is displaced from solution and absorbed by the mercury. Sodium is fed at the rate of 90 lb. per day, and silver amalgam is withdrawn at a rate varying from 35 to 350 lb. per hour. Barren solution leaving the apparatus has a gold and silver value of $0.07 per ton, and contains 4.1 lb. free NaCN and 1.7 lb. equivalent $Ca(OH)_2$. The efficiency of the process is 99 per cent, since the gross value of the precious metals in the feed, with silver at 60c. and gold at $20 per ounce, was $7. The process is equally applicable to the recovery of copper from cyanide solutions. (1,164,636, Dec. 21, 1915.)

Zinc

Hydrometallurgy of Zinc with Electrolytic Deposition.—An insight into the methods to be used by the Anaconda Copper Mining Co. in the wet treatment of zinc ores and production of electrolytic zinc at Great Falls, Mont., may be had from patents recently granted to FREDERICK LAIST and FREDERICK F. FRICK, of Anaconda, Mont., members of the technical staff of that company.

The invention provides a cyclic process specially adapted to the treatment of complex sulphide ores containing such metals as lead, iron, copper, silver, gold, etc. The proper performance of the cycle depends on the presence of manganese, which may either be a normal constituent of the ore or added in the form of manganese carbonate or sulphide. The addition may be made either on starting or after the operation has been established. The function of the manganese is to serve as a carrier of oxygen to oxidize ferrous compounds to ferric, and thus make possible the complete separation of iron from solution.

Sulphide ores or concentrates are first roasted. Manganese is introduced subsequent to the roasting operation, and in such quantity as may be required by the ferrous iron to be oxidized. After roasting, the ore is leached with sufficient dilute sulphuric acid so that there is from 5 per cent to 10 per cent excess acid over the amount required to complete the reaction. The oxidized metals go into solution as sulphates, some of the iron as ferrous sulphate. In this state the iron is not removed by adding lime or zinc oxide, and it is for the oxidation of this ferrous iron that manganese in its higher forms of oxidation is used. It may be present in the form of manganese dioxide or permanganic acid and possibly other states of oxidation higher than the manganous. It may be present either in solution or suspension, according as it is soluble or insoluble. In any event in these higher states of oxidation the manganese is a product of the final electrolytic stage of the process, and is returned to the proper stage for the oxidation of ferrous iron. Having oxidized all the iron, that metal can now be precipitated by milk of lime of zinc oxide. Iron, arsenic and antimony are thus precipitated. For the removal of copper from the electrolyte, and possibly small amounts of silver, arsenic and antimony also, zinc dust is added to the solution. At the stages of precipitation of the various impurities, decantation or filtration of the solution follows, and finally a pure electrolyte of zinc sulphate is obtained, containing manganese. This solution is then electrolyzed, using insoluble anodes, as of lead, and cathodes of zinc starting-sheets. The products of electrolysis are zinc, sulphuric acid and manganese dioxide and permanganic acid. The solution containing the sulphuric acid and manganese is now available for further leaching, oxidizing the ferrous iron and facilitating its removal. According to a modification, an excess of the roasted ore may be used for precipitating iron, arsenic and antimony; but since ferrous iron might also be added in this procedure, a certain amount of manganese dioxide derived from electrolysis is introduced at the same time. Since the roasted ore is used in excess for this precipitation, a small residue would remain, and this is to be mixed with raw ore in the roasting furnace, at which time the ferric hydroxide in the residue is converted largely to ferric oxide, which is insoluble, while the zinc is unaffected and may be dissolved. (1,167,700-1, Jan. 11, 1916.)

Removal of Cadmium from Zinc Ores.—In view of the greater value of spelter free from cadmium, for certain commercial purposes, it is desirable to free cadmium-bearing zinc ores from this impurity before smelting them for their zinc content. Mr. GILBERT RIGG, of Palmerton, Pa., proposes to accomplish this by heating the ore to a temperature of from 700 deg. to 850 deg. C. in an atmosphere of a reducing gas, such

as producer gas. The process may be performed in an inclined rotary cylindrical kiln, heated internally by the combustion of the producer gas. The ore enters at the upper end and discharges at the lower. Provision is made for regulating the supply of gas and air, so that when the temperature of the furnace tends to decrease, the supply of air may be increased to restore the higher temperature. In fact it may be found desirable to operate the process with alternately high and low temperatures within the range indicated.

According to a patented modification of the process, the ore may be supported on a suitable grate, either stationary or traveling, within a furnace structure, wherein provision is made for a down-draft through the mass. In such case the fuel would be on top of the layer of ore, and as the heat penetrated downward the cadmium would be volatilized and drawn off in the gases. Instead of placing solid fuel on top of the ore, an ignited mixture of air and producer gas could be drawn downward through the bed and the temperature be alternately increased and decreased within the indicated limits until the cadmium was removed. (1,161,885-6, Nov. 30, 1915.)

Aluminium

Production of Aluminium from Clay and Other Aluminium Silicates.—Fig. 2 illustrates the flow-sheet of a process for obtaining aluminium from its silicate combination, the method being patented by GRENVILLE MELLEN of East Orange, N. J. The clay or kaolin is fused with sodium sulphate and sulphuric acid, or with its equivalent of sodium bisulphate, in proportions to form aluminium sulphate and free silica. After the reaction is completed the mass is cooled and dissolved in hot water or in a hot dilute solution of sodium sulphate containing a small quantity of aluminium salts from a prior extraction. The hot solution thus obtained

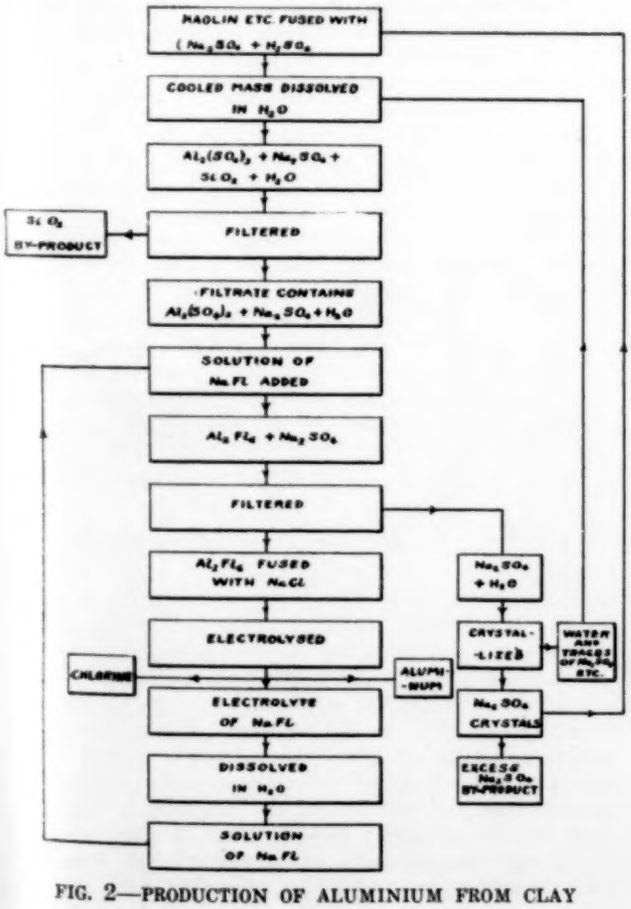


FIG. 2—PRODUCTION OF ALUMINIUM FROM CLAY

is filtered and the filtered concentrated if necessary. At this stage a concentrated solution of sodium fluoride is added, precipitating aluminium fluoride, which is separated by filtration. The filtrate containing sodium sulphate is concentrated to crystallize the contained salt which is again used in the process. The aluminium fluoride is fused with sodium chloride and electrolyzed, producing aluminium. By-products are pure silica, chlorine and sodium sulphate. The process is cyclic, as seen by the flow-sheet. (1,160,431, Nov. 16, 1915.)

Melting Aluminium.—A new process for melting aluminium or its alloys, having for its object the protection and preservation of the metal from oxidation and the production of sound castings, is patented by GRENVILLE MELLEN of East Orange, N. J. In carrying out the process the metal to be melted is placed in a crucible in an air-tight chamber. The air is then removed from the chamber and an inert gas admitted. After the metal is melted in this inert atmosphere, and while the metal is yet molten, the inert gas is removed from the chamber, thus producing a partial vacuum therein. Occluded gas is thus removed from the molten metal, and it is then ready for pouring, which should be done as quickly as possible. (1,160,430, Nov. 16, 1915.)

Vanadium, Uranium and Radium

A process for extracting vanadium, uranium and radium from ores is patented by RICHARD B. MOORE of the United States Bureau of Mines, Denver, Col. The process is that which has been used by the National Radium Institute, Denver, and which has been referred to in this journal, Dec. 15, 1915, page 965, and Jan. 1, 1916. It is also the subject of *Bulletin* No. 104, Bureau of Mines. Reference is here made to the patents to record the fact that they are dedicated to the public and can be used without payment of royalty. (1,165,692-3, Dec. 28, 1915.)

Electrolytic Processes

Electrolytic Cell for Treating Ores.—A revolving electrolytic diaphragm cell for the hydrometallurgy of ores is patented by LEWIS E. PORTER of Los Angeles, Cal. (one-half assigned to Hugh E. Stock of Casper, Wyo.). The cell consists of two concentric cylinders, the inner one forming the cathode and being made of metal and the outer one made of wood and provided on its inside surface with a number of plates which act as anodes. These plates are connected to an outside slip ring and the cathode is also connected to a slip ring. The cylinders are both on the same shaft and the outer cylinder is geared to a driving pinion for revolving the cell. Surrounding the inner cathode cylinder and close to it is placed another wooden cylinder constituting a diaphragm containing a number of small perforations and over this a filter cloth is placed. The action of the cell is as follows: The anode chamber, which is the outside annular space, is filled with ore through a suitable opening and electrolyte nearly saturated with metallic salt is forced in under pressure, filling any spaces not filled by the ore and being forced through the diaphragm and filter cloth into the cathode space between the diaphragm and inside cylinder. The ore in contact with the plates on the inside surface of the outer cylinder forms the anode and solution takes place, with metal subsequently depositing on the cathode. The cathode cylinder has horizontal wooden strips on its surface which divide the deposited metal into several strips, making it more easily removable. Suitable pipes are arranged for conducting away any gases formed in the anode or cathode chamber and the anode chamber pipe is provided with a safety blow-off valve, in case of too great a gas pressure developing.

The cell is stated to be applicable to the recovery of zinc from zinc ores containing considerable iron. (1,167,594, Jan. 11, 1916.)

Non-Metallic Cathode for Sodium Chloride Electrolysis.—A cathode for use in diaphragm cells which is intended to prevent the deposition of calcium or magnesium salts, found as impurities in salt solutions is patented by FRANK MCDONALD of Roaring Springs, Pa. This cathode is made of graphite and a form of cell illustrating the use of this cathode is shown in Fig.

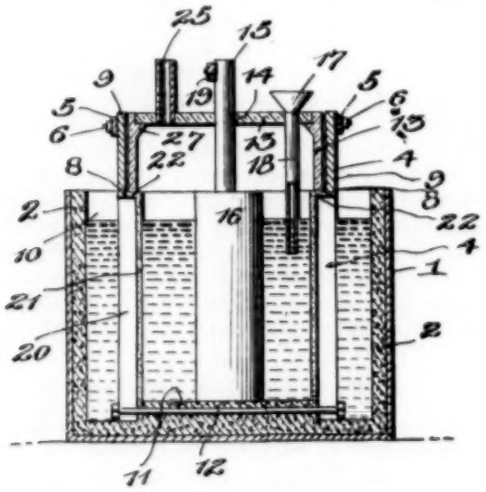


FIG. 3—CAUSTIC SODA-CHLORINE CELL

3. The tank or cell is cement-lined and the diaphragm is shown at 21 and the perforated graphite cathode at 20. The anode is shown at 16 and chlorine space inclosed above the anode space by slate cover 13 fastened to the cathode and diaphragm. Chlorine is conducted away through pipe 25 and caustic soda overflows from the outside cathode chamber through an outlet in the side not shown. The cathode and diaphragm are clamped to the raised cement projectory 11, and the cathodes are made porous by saw cuts in that portion which is below the solution. The diaphragm is of asbestos sheets. It is claimed that this cathode does away with the objectionable features of metallic cathodes in the use of which the internal resistance may rise owing to the deposition of non-conducting salts of calcium and magnesium existing as impurities in the salt solution, which would also clog the perforations of the cathode. (1,167,705, Jan. 11, 1916.)

Bismuth

Method of Purifying Bismuth.—A series of treatments forming a process for the production of pure bismuth is patented by WALTER C. SMITH of Chrome, N. J. (assigned to U. S. Metals Refining Co.) In the treatment four separate operations are performed as follows (not necessarily in the order given): 1. Sulphur is added to the molten impure bismuth metal while agitating and the temperature lowered to 515 to 520 deg. Fahr. By this treatment all of the copper and the greater portion of the tellurium, silver, arsenic, antimony, selenium and zinc, some of the tin and a fair portion of the lead are removed. The impurities form a matte which floats on top and may be removed. 2. Caustic soda is added to the molten metal and agitation commenced by steam. The caustic removes the tellurium, selenium, sulphur, arsenic, antimony, tin and zinc, forming sodium compounds which float on the surface. Silver and gold will be unaffected. 3. Zinc is added to the molten metal while agitating and the temperature lowered to about 550 deg. Fahr. This further reduces the amount of silver present and removes all the gold. The remaining lead and practically all the remaining silver and any gold remaining are removed by crystallization. The steps may be modified or rearranged to suit conditions. (1,166,721, Jan. 4, 1916.)

Alloys

Electric-Resistor Material.—An alloy of nickel, copper and chromium to be used for electrical resistors is patented by MATTHEW A. HUNTER of Troy, N. Y. The alloy is stated to combine high resistivity with low temperature coefficient of resistance. The proportion of nickel used is large, relative to the copper and chromium and an alloy which has been found to give good results consists of the following: 85 parts nickel to 15 parts copper, with 20 parts chromium to 100 parts of nickel and copper. This alloy has a resistivity of 113 microhms per centimeter cubed at 20 deg. C., and at that temperature has a temperature coefficient of resistance of 0.000078 ohms per deg. C. per ohm. The copper increases the resistivity up to a certain point, and the chromium effects a reduction in the temperature coefficient of resistance. (1,168,074, Jan. 11, 1916.)

Copper Alloy.—An alloy consisting of copper, aluminium, antimony, iron and silicon is patented by WILLIAM C. PEASE of Marietta, Ohio. In making this alloy aluminium is first heated in a crucible, and while it is melting a piece of clay about the size of an egg per pound of aluminium is added. When this mass has thoroughly melted iron pyrite is added and stirred well. This is then poured off and broken into small pieces. One ounce of this mixture is added to 1 lb. of molten copper. The proportion of aluminium to iron pyrite is usually 16:2, although this may be varied. For further hardening the copper, antimony may be added. An average analysis of this alloy would be: copper, 88.4; aluminium, 6.6; antimony, 1.26; iron, 1.96; silicon, 1.60. It is claimed that the alloy may be molded, machined or forged. (1,168,962, Jan. 18, 1916.)

Electroplating

Continuous Electroplating Apparatus for Magnetic Material.—A continuous plating apparatus, designed primarily for electrogalvanizing pipes, rods, or other long, slender magnetic metal articles, is patented by NEWTON W. BUCH of New Castle, Pa. (assigned to Safety Armorite Conduit Co. of Pittsburgh, Pa.). A cross-section of this apparatus is shown in Fig. 4. The

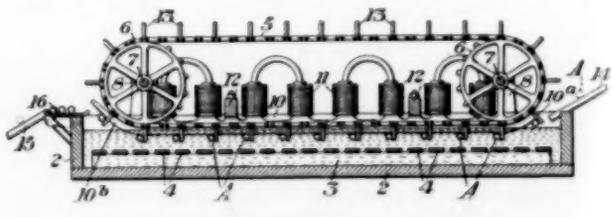


FIG. 4—ELECTROPLATING APPARATUS

tank 2 may be made of any suitable width to suit the work to be done. The electrolyte 3 fills the tank as shown and anode members 4 are arranged toward the bottom of the tank. An endless chain having protruding pieces 13 fastened to it revolving on sprockets as shown passes through the solution. Below this endless chain or belt and in the solution is cathode member 10, which is turned up at the ends in order to facilitate introduction and delivery of the material to be plated. The cathode members, one of which is placed parallel to the one shown on the other side of the tank, form pole pieces for electromagnetic coils shown at 11 . 12, 12

are cross-bars for supporting the cathodes and magnetic coils. The material to be plated is introduced at 14, is passed through the solution by the moving chain while being attracted to the cathode on account of its magnetic force. The material is delivered at 16 by the protruding members of the traveling belt. (1,168,281, Jan. 18, 1916.)

Electroplating Apparatus for Plating Pipes.—An apparatus for plating pipes and other magnetic material is patented by NEWTON W. BUCH of New Castle, Pa. (assigned to Safety Armorite Conduit Co. of Pittsburgh, Pa.). This apparatus consists of a large cell having stationary anode plates and cathode racks or lifting frames consisting of a number of magnetic coils, suitably spaced on parallel bars and forming a coherent unit which may be picked up by a crane or other lifting device. The articles to be plated are placed in a rack, the magnetic lifting frame is placed on them by the crane and the current turned on in the magnetizing coils. The articles to be plated are thus raised up and conveyed to the cell or tank where they are lowered into place and disengaged from the crane. The magnets still hold on to the articles while they are being plated, and when finished they are again removed and replaced in their former rack by releasing the current in the magnet coils. The advantages claimed are that rapid and efficient work can be done and in large quantities at a time, and the articles can be plated all over, as no mechanical holder covers up any space. (1,168,280, Jan. 18, 1916.)

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver

Effect of Heat in Cyaniding Gold Ores.—The literature on this subject is scant and the data too general. In order to make a more careful investigation of the subject, Mr. E. A. WRAIGHT has performed several series of experiments which he has recorded in *Bulletin* 135, Inst. Min. & Met. The first tests were made on a silicious gold ore containing about 7 per cent pyrites, and 4 oz. gold and 5 oz. silver per long ton. This ore was ground to pass a 60-mesh screen. It was agitated in duplicate quantities in hot and cold cyanide solution for definite periods. In the same manner a basic ore containing limonite and about half the gold and silver as the first ore. The ore was ground to pass 120-mesh. The results obtained from these ores indicated that agitation in a hot solution was of very doubtful benefit. In the first case the hot solution showed a better extraction than the cold for the first hour, but after that the hot solution was at a decided disadvantage. In the second case similar results were obtained, the hot solution giving better extraction for the first hour, but only equaling that of the cold solution at the end of four hours. In both cases, however, the consumption of cyanide was higher in hot solutions. In explanation of the results the author cites the fact that the solubility of gold in cyanide solution increases up to a certain temperature, beyond which there is a decrease. Further, the solubility of minerals like pyrite, in cyanide solution, increases with the rise in temperature, and probably the increased solubility of base minerals counterbalances the solubility of the gold, resulting in a greater loss of cyanide and a consequent decrease in gold dissolution.

Further experiments were made with heated air for agitation, using air at a temperature of about 350 deg. C. The results of these tests showed that while the hot air accelerated the rate of extraction, it did not give as high a total extraction as under cold conditions. The acceleration noted is probably due to the increased activity of the oxygen, an idea which is apparently confirmed by experiments with hydrogen peroxide added to cold solution to supply oxygen in an active form. Other oxidizing agents, such as potassium bichromate and permanganate, were tested, with decidedly deleterious action.

The author concludes:

1. The effect of heating cyanide solutions is of very doubtful benefit; the extraction may be increased for a short period, but this is more than compensated by the increased cyanide consumption and the subsequent decrease in the rate of dissolution of gold.

2. Oxidizing agents (H_2O_2 excepted) are apparently of no value and may even exercise a deleterious effect on the extraction.

3. The addition of oxygen in a more active form, either as hydrogen peroxide or by means of heated air, increases the solvent activity of cyanide solutions in a very pronounced manner.

Copper

The Role of Sulphur in Pyritic Smelting.—In the course of his address as president of the Australasian Institute of Mining Engineers, Mr. Robert C. Sticht made the following remarks relative to the fuel value of sulphur in smelting at Mount Lyell. The complete address appears in the Institute *Proceedings*, No. 19, 1915:

"On its downward course the 'pyrrhotite' suffers a further desulphurization in the preparatory zone, in the hot current of neutral gases, under the influence of increasing temperature. It progressively loses more and more of its sulphur, until the particular combination of iron and sulphur, which finally reaches the focus, is even lower in sulphur contents than the monosulphide, FeS. Its composition naturally greatly varies, but for practical purposes it may be regarded as a solution of a little metallic iron in the monosulphide. Empirical formulæ, like 4FeS, Fe, and 3FeS, Fe, are commonly indicated as representative of this residual sulphide when the working of the furnace is investigated by calculation.

"I do not, however, regard this high desulphurization as an advantage, but rather as a defect of our work—not on account of the possible liberation or precipitation of metallic iron in quantity, and the formation of 'sows' (a phenomenon which has never been observed at Mount Lyell), but because the particular sulphur and iron compound which thus reaches the focus is the one that does all the work there, and, under the circumstances, the shortage of even the small amount of sulphur involved is a drawback. It has not yet been possible to regulate the composition of the sulphide that burns in the focus, and it is possible that, if we could currently burn a composition more like FeS there, the small amount of coke which it is still necessary to use to furbish up the general furnace heat might be curtailed, or altogether avoided. The point seems rather fine, and the endeavor savors of splitting hairs, but it is certainly inaccurate to take it for granted, as is usually done, that it is FeS that burns in the focus, when it is actually a less energetic compound.

"From the foregoing it will be obvious that the sulphur in the original pyrites does not contribute very vitally to the heat developed in the pyrite furnace, notwithstanding its great quantity. The Mount Lyell pyrites now commonly has about 46 per cent of sulphur, and the North Lyell ore has 7.5 per cent. The ordinary furnace mixture of the two has about 30 per cent S.

Yet, of the original total S in the FeS_2 treated (*i. e.*, not counting the sulphur protecting the copper in its passage through the furnace), only 36-38 per cent actually is burnt in the focus as a fuel. A bare 2 per cent of the total goes into the matte, and from 55-60 per cent of the original S in the FeS_2 distils out of the furnace as sulphur vapor, without combustion anywhere, except in the shaft above the column, where it comes into contact with the atmosphere. The temperature at the throat is high enough to ignite it. These figures indicate the loss of a very considerable portion of the total inherent fuel value of pyrites.

"Unfortunately, there is no hope of realizing the calorific value of the sulphur thus escaping from the depths of the furnace, for the volatility of the vapor cannot be restrained, and the preferential affinity of the oxygen in the blast for the iron also discriminates against the combustion of the metalloid. Of the metal, iron, the furnace oxidizes quite regularly 95 per cent of the total contents in the ores. By contrast this represents a very valuable utilization of a constituent which is present in smaller quantity, and also has a lower calorific value, than the sulphur, for it saves the situation.

"There is, similarly, no prospect of utilizing any portion of the loose (*i. e.*, volatile) atom of sulphur out of the pyrites, which at once sublimes off at the throat. The suppression of all these eliminations would only be possible if the tension of the furnace atmosphere could be sufficiently increased to counteract the vapor tension of the subliming sulphur. This, however, is beyond the reach of our pressures, and impracticable in our apparatus.

"Were it feasible to burn the original FeS_2 in its entirety to the same degree as the Fe alone is now burnt, then the cokeless smelting would be accomplished. But it would have to be done in an apparatus differing very considerably from our present blast-furnace (or converter), and it is not easy to forecast the constructional lines of a closed, high-pressure structure, which it would have to show. It would have to embody all the accessibility of the throat for feeding and barring which the blast-furnace presents, together with a sealed tunnel-head, and at the same time be as simple to handle as our blast-furnaces now are. On the other hand, the mere raising of the blast-pressure in the existing furnaces will not lead to the combustion of noticeably more of the sulphur. Unless the ore column is simultaneously increased, a very heavy blast will only cause a waste of wind, by escape from the top, and the formation of a low-grade matte, owing to the lack of preparation of the descending masses in consequence of the rising of the focus. Some of the escaping sulphur may be burnt in such a case, but this will be above the stockline. Such back pressure as can be arranged for in the present furnace construction by means of tight charging-doors, etc., is insufficient to keep down the sulphur to any extent, and inadequate to force it to remain fixed until it gets to the focus in greater proportion.

"Something was hoped for in this general connection from the increase in height of the Mount Lyell furnaces. The column used to be 10-12 ft., and may now be carried up to 18 ft. No palpable effect has so far resulted from our efforts to cut down the coke and burn proportionally more of the sulphur. It must, however, be explained that just about the time the column was raised the percentage of Fe and S in the Mount Lyell pyrites began to decrease, while the Zn and Pb increased, and this change of composition has since become permanent. It has had an unavoidable deleterious effect on the heat developed by the charge alone, and, instead of being able to lower the coke, we have been obliged to raise it. We are satisfied, however, that, with the former lower column, we would be obliged to use still more coke, for the same blast volume."

Crushing and Grinding

The Dumb-Bell Tube Mill.—In the *South African Mining Journal*, Nov. 13, 1915, Mr. G. A. ROBERTSON describes a new type of tube mill. Instead of using pebbles as grinders it contains a roller of dumb-bell shape. The mill tested was 3 ft. 6 in. long and 2 ft. 9 in. in diameter. The roller has its weight concentrated at the two ends, and each end has a concave face which fits two convex faces in the shell of the mill. As motion is imparted to the mill the roller revolves, and when ore is fed it becomes ground between the faces described. Both ends of the mill are the same, and it can be fed and discharged from both ends simultaneously. The dumb-bell mill is equipped with centrifugal classifiers fitted into the trunnions at each end. These classifiers revolve with the mill, and all particles of sand which drop into the pockets of the classifier are returned for further grinding. The following is a record of two tests made at the May Consolidated, using a sand feed of + 0.0249 in.:

Mesh	Intake, Per Cent	Discharge, Per Cent	Mesh	Intake, Per Cent	Discharge, Per Cent
+ 30	3.5	—	+ 30	7.3	—
+ 60	29.5	0.5	+ 60	33.3	12.3
+ 90	19.2	22.9	+ 90	16.8	28.4
+200	12.5	22.8	+200	10.9	14.7
—200	35.5	53.8	—200	31.7	44.6
	100	100		100	100
	Moisture, 92.3 per cent. Duty per 24 hours, 5.2 ton.			Moisture, 90.4 per cent. Duty per 24 hours, 6.5 ton.	

The following figures show the mill's performance when fed at both ends and discharging at both ends. The feed passed a screen of 0.03-in. aperture:

Test	Screen	Per Cent Inlet	Per Cent Outlet	Moisture in Feed Sample, Per Cent	Duty Per 24 Hours, Tons
(1)	+ 60	31.4	11.1	75	16.5
	+ 90	16.5	24.1		
	— 90	52.1	64.8		
(2)	+ 60	24.7	15.6	Full	
	+ 90	16.7	22.0	battery	25
	— 90	58.6	62.4	water.	
(3)	+ 60	42.1	30.2		
	+ 90	21.5	27.0	60	13.5
	— 90	36.4	42.8		
(4)	+ 60	42.8	16.9		
	+ 90	18.7	33.4	75	14.5
	+200	12.2	16.7		
	—200	26.3	33.0		

In order to determine the performance of the mill with coarse feed, it was fed with material — ½ in. + 0.03 in. with the following results:

	Inlet	Outlet
+30	67.5	15.5
+60	18.5	40.5
+90	5	15.5
—90	9.0	28.5

Duty 7 tons per 24 hours.

It is estimated that 4 hp. will drive this 3-ft. 6-in. by 2-ft. 9-in. mill at 40 r.p.m. The capital expenditure is small and the wear is confined to the convex and concave faces of the shell and roller. The water line is practically a horizontal plane. Grinding is done both by impact and compression. The designer believes that the size of feed can be increased as the weight of roller increases.

The **Mojave Tungsten Company**, recently formed, is preparing to operate two developed, four partially developed, and several undeveloped claims in San Bernardino County, California. A modern concentrating mill will also be erected which may be used for custom purposes as well as for the company's own use. Several well known men are identified with the new company.

High Temperature Insulation

A Paper Read Before The Chicago Section of the American Society of Mechanical Engineers on January 14, 1916

BY P. A. BOECK

The problems of conservation of heat and prevention of thermal losses have been the subject of considerable investigation and research in certain industries, more especially in the application of cold-storage installations for low temperatures, and of steam lines and electric furnaces in high-temperature work. It has not been until recently, however, that this subject as applied to general industrial purposes, has received the attention of designers and engineers that its importance deserves.

Heat flow is a rather difficult factor to measure under the working conditions of an industrial furnace. It was not until the comparatively recent introduction into certain operations of the heat balance sheet, with a systematic attempt to account for the discrepancies in the totals, that the various causes of heat losses and the reasons for their existence were brought to light. Moreover, such rapid advancement and radical changes in the design of equipment, especially in metallurgical lines, have occurred that practice has apparently outdistanced the theory of design. In the modern tendency toward greatly increased size of units, effort has seemingly been made to utilize the heat of fuel gases without proper attention having been paid to conserving heat energy and confining it strictly to the points of maximum activity.

The work of several of the Government Bureaus, notably that of Bureau of Mines and Bureau of Standards, and the systematic investigation of the thermal properties of structural materials by a number of the technical societies are forming the basis for more intelligent and consistent work in the economically important subject of the prevention of fuel waste. We would refer to the recent excellent paper of L. B. McMillan[1] before this society. Here the point is made that it is not alone the efficiency of the insulator nor its initial cost, but a factor consisting of both of these items, together with the factor of durability or permanence, which form the ultimate criterion of the most effective insulator for any specific purpose.

[1] L. B. McMillan: The Heat Insulating Properties of Commercial Steam Pipe Coverings, Transactions of the American Society of Mechanical Engineers, 1915.

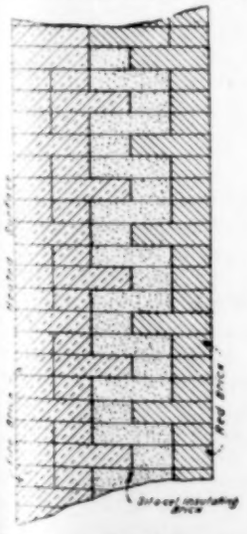


FIG. 1 — TYPICAL WALL CONSTRUCTION WITH SILOCEL BRICK LAID IN WALL

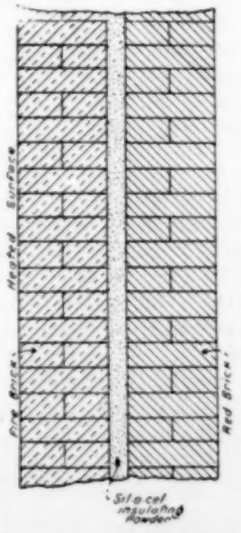


FIG. 2 — TYPICAL WALL CONSTRUCTION WITH SILOCEL POWDER IN HOLLOW WALL

It should be unnecessary in this practical and utilitarian age to call attention to the need of preventing heat losses, but the consistent and intelligent application of insulation to high temperatures has brought out other advantages in addition to the saving in fuel. The most important of these advantages from an industrial point of view, is that of increasing the furnace capacity, it not being uncommon to add as much as 10 to 15 per cent to the furnace capacity without increasing the size of the equipment, the fuel consumption and overhead expense.

This is more evident when dealing with higher temperatures, considering that no matter how much fuel is consumed to produce a given product, the results are not obtained until the temperature attained is high enough to bring about the reaction. Any increase in the temperature and heat capacity of the equipment above that necessary to get results, produces an increase in output equivalent to many times the actual temperature difference.

The insulation of hot-blast stoves, for instance, is being advocated for the heat saved. This is a large and important factor which has been frequently brought out, but it is of minor importance when compared with the advantages of increased tonnage produced. Therefore, when the reference is made in this discussion to heat saving and increased capacity, we have in mind the effect of adding increased energy at the point of maximum effectiveness without additional fuel.

ADVANTAGES OF INSULATION

In any high-temperature apparatus if the heat can be prevented from being dissipated from the outside of the furnace, the temperature of the furnace is made more uniform and the zone of the reaction and consequently the capacity, are manifestly increased without additional fuel consumption. Furthermore, overheating at the source of heat in the effort to bring as much of the furnace as is possible into the zone of reaction, is materially reduced by preventing the loss through the furnace walls. This has the ultimate effect of lowering the temperature at the source of heat, greatly increasing the life of the refractories and the capacity of the furnace.

It is evident also, that in furnaces which are surrounded by metal casings, deterioration and damage caused by overheating may be many times as great as

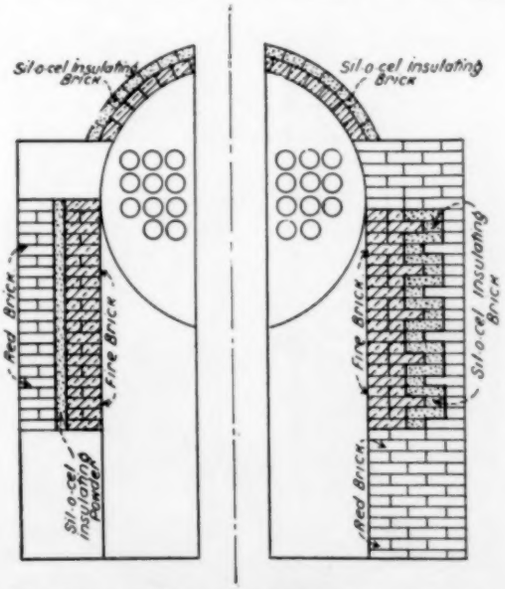


FIG. 3—INSULATION OF BOILER SETTING BY MEANS OF SILOCEL BRICK AND POWDER

the cost of installing the proper insulation. The advantages gained by the protection of workmen from unhealthy and unsanitary conditions, incident to their laboring under over-heated conditions, are also apparent.

In general then, the question of insulation does not concern itself so much with the saving in fuel, which, of course, is an important item, as with *the general effect produced by confining the heat generated to points of maximum effectiveness.* The higher the cost of fuel, the more necessary it is to look closely to the preventable losses to gain an economically operated furnace.

HEAT FLOW

Inasmuch as the amount of heat conducted by a body is one of the principal sources of loss in high-temperature equipment which we have under consideration, it will be of interest to inquire into this in order that we may control the flow of heat by this method as far as possible. Heat is a form of energy consisting of molecular vibration of a periodic character, and subject to the general laws of wave motion. It can be reflected, refracted and dispersed by creating the proper conditions. In other words, by the proper mechanical manipulation we can increase or decrease, within certain limits, the amount or rate of heat flow. One of the most common ways of doing this is by introducing alternate layers of materials having different heat-transmitting values, thus breaking up or changing the wave length. The introduction of air cells or voids within the body is a most effective method of accomplishing the reduction of the conductivity. In fact, the apparent density of a body (which indicates the proportion of voids present) may be taken as an approximate criterion of its heat conducting capacity, i.e., the body is made more opaque to the propagation of heat vibration. This, however, is more apparent where the size of the voids are very small and the number great. Large voids have the effect of propagating heat by convection and radiation as will be seen later.

Heat is transferred from one place to another by three different methods, conduction, radiation and convection:

Conduction.—Conduction is the action taking place in the transfer of heat in which the heat energy is passed along from a particle at a higher temperature to a lower one by virtue of their contact. While this method of heat transfer applies to the three states of matter, it is only of magnitude in solids.

Radiation.—By radiation heat is transferred from one body to another without any material agency by wave motion; this applies only to the gaseous state of matter.

Convection.—Convection is that process of heat transfer in which some portion of the body whose temperature is raised, moves to another place in the medium where the temperature is lower, thus tending to raise the temperature of the medium. This method of heat transfer is therefore limited to liquids and gases.

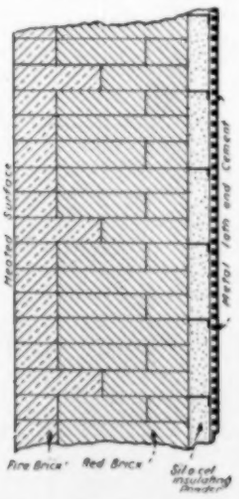


FIG. 4 — TYPICAL WALL CONSTRUCTION SHOWING SILOCEL POWDER SUPPORTED BY METAL LATH, CEMENT FINISH

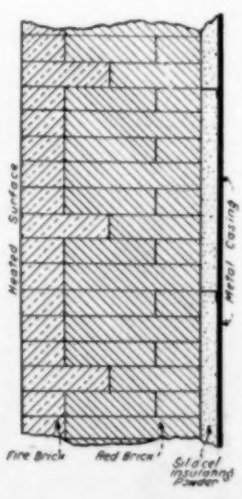


FIG. 5 — TYPICAL WALL CONSTRUCTION WITH SILOCEL POWDER SUPPORTED BY METAL CASING

STEAM BOILER

This is illustrated by the well known example of a steam boiler.[3] The heat generated by the fuel burning in the firebox is imparted to the dry surface of the boiler plate by radiation from the hot fuel bed, furnace walls and luminous flames, and secondly from the moving gaseous products of combustion by convection. From the dry surface it travels through the soot, metal and scale to the wet surface purely by conduction and is in turn transmitted into the boiler water mainly by convection.

The energy in the form of heat which eventually finds its way into the water in the boiler is, of course, subsequently available in other forms. That portion, however, which is transferred to the walls of the furnace setting by any of the three methods mentioned, is conducted through the setting and lost by radiation or convection from the outer surface of the setting, if means are not provided to prevent. It is the heat lost in this manner and methods for its prevention that will be especially considered in this discussion.

The selection of a boiler setting to illustrate the various methods for heat propagation was taken as that most familiar to general practice, but the selection is, unfortunately for the illustration, one in which the heat losses through the settings are comparatively low.

RATE OF HEAT FLOW

The rate of transfer of heat under various conditions, by both conduction and radiation, shows the relative importance of these two methods in influencing heat losses.[5] The amount of heat conducted through a unit area from one part of the body to another is proportional to the difference in temperature of the two parts; directly proportional to the thermal conductivity of the body through which the heat passes and inversely proportional to the distance between the two parts of the body. In other words, the conduction of heat through a solid body from one place to another is a direct function of the conductivity of the body and the difference in temperature of the two planes, and an indirect function of their distance apart.[4] This is identical with Ohm's law for transfer of electrical energy.

RADIATION

The heat transferred from one body to another by radiation is proportional to the difference of the fourth powers of the absolute temperatures of the two bodies. While this is strictly true only of the ideal "black-bodies," the variation is so small that for all practical purposes, this relation holds good in ordinary procedure.[6]

HOLLOW-WALL CONSTRUCTION

This relation indicates why a hollow-wall space is an effective insulator in low-temperature work, such as re-

[3]Kreisinger, Henry, and Ray, Walter T.: The Transmission of Heat into Steam Boilers, Bulletin No. 18, U. S. Bureau of Mines (1912).

[4]Convection and Radiation of Heat. Irving Langmuir. Transactions Am. Electrochem. Society, Vol. 23, p. 299 (1913); Metall. & Chem. Eng'ing, Vol. II, p. 278 (May, 1913).

[4]Ray, Walter T., and Kreisinger, Henry: The Flow of Heat Through Furnace Walls, Bulletin No. 8, U. S. Bureau of Mines (1912).

[5]Gurney, Harold P.: Heat Radiation, *Journal of Ind. and Eng. Chemistry*, Vol. III, No. 11, p. 807, Nov., 1911.

frigeration, etc., whereas in high-temperature operations the loss of heat by radiation through a hollow-wall space is so great that its insulating effect is less than if this wall space were filled with material of rather high thermal conductivity. This has been brought out by Ray and Kreisinger who demonstrate that the hollow-wall space type of wall construction is much less effective as a means for preventing the loss of heat than a solid wall of any ordinary construction material of equal thickness. This is especially true if the air space in the hollow wall is near the furnace side and becomes highly heated. This is entirely contradictory to the general belief that, since air is a poor conductor of heat, air spaces built into the walls of a furnace will greatly reduce heat loss by radiation. While the heat does travel very slowly through the air by conduction, it leaps over the air space readily by radiation, because the quantity of heat which passes across the hollow space is a function of the fourth power of the absolute temperatures of the surfaces inclosing it, which loss is enormously increased by rise in temperature.

FURNACE WALLS

In general, in high-temperature furnace construction, there are two separate and distinct factors, which must be considered to produce an effective wall.

The first of these is to provide a material having the ability to resist the action of high temperatures, sufficient mechanical strength, and, possibly, the property of resisting corrosive slags, gases, etc., without spalling or being eroded.

Secondly, to prevent the excessive loss of heat due to conduction from the interior of the furnace to the outside where it is lost.

It is rare that a good refractory material is an insulator; usually it is necessary to augment or back up the refractory with some material having a much lower heat-conducting capacity.

REQUIREMENTS OF INSULATORS

The requirements of the insulating backing for the more refractory lining are rather severe. An ideal material would have the following properties: It should have an extremely high insulating value, and be sufficiently refractory so that no fusing or excessive shrinkage would take place in that portion which is in direct contact with the highly heated refractory wall. It should not be decomposed or be changed greatly in volume at the temperature, and it should, furthermore, be of light weight, unaffected by moisture, of convenient form, readily applied by unskilled labor, and low in cost. It should be of such composition as not to react upon or attack either the refractory material or the metal shell of the container, even in the presence of moisture; i. e., it should contain no free acid radicals, and should not be broken or caused to settle by vibration or heat. It should not have high expansion and it should be sufficiently elastic to take up strains between the lining and the shell produced by temperature changes.

While this is a rather formidable array of requirements, there are products upon the market which fulfill practically all of them. Such materials as are being commonly applied to steam-temperature insulation, of which magnesia and asbestos are best known, are, of course, eliminated from consideration in the great majority of cases mentioned here on account of decomposition, shrinkage and settling. The lack of more extended practice in high-temperature insulation must be attributed to the fact that effective high-temperature non-conductors have not been generally available.

AMOUNT OF INSULATION

When the kind of refractory material that is best suited to the surface has been determined, the next most important item is the thickness of the walls and the nature of the insulating material most suitable, and it is necessary to determine the degree of insulation which will produce the most effective results.

Hering has pointed out that, in order to effect a perfect insulation, the furnace should be surrounded with an insulating material that would maintain the same temperature at any point as that of any adjacent point of the lining.[6]

In attempting to obtain perfect insulation, it is entirely possible to over-insulate, causing a serious damage to the refractories in the high-temperature zones of the furnace. To maintain the inner walls, it may be necessary to permit a considerable flow of heat through the wall, with a corresponding decrease in the temperature of the refractories at the heated point, in order to prevent their destruction.

FIG. 6 — HOT-BLAST STOVE INSULATED WITH SILOCEL POWDER AND BRICK

It would be manifestly impracticable to insulate the roof of an electric steel furnace or open-hearth furnace, for instance, with a heavy layer of insulating material, because the cost of the refractory lining which would be destroyed would be considerably greater than the cost of the heat which would otherwise be lost.[7]

However, the issue in some instances may be met by other practical methods such as increasing the working capacity of the unit in order to make use of the additional heat available, rather than to let this valuable heat at the critical period escape to save the refractory lining. In calcining celite, the product which we will consider further in this paper, the temperatures are controlled entirely by the working load imposed on them without allowing the oil burners to ever fall below their lightest working capacity and highest heat. To produce celite calcined to 2500 deg. F., an average kiln has a daily capacity of about 20 tons. When celite is wanted which has been calcined at 2000 deg., the kiln is pushed

[6]Hering, C.: Thermal Insulation of Furnace Walls, *Metallurgical and Chemical Engineering*, vol. x, No. 2, p. 97 (Feb., 1512).
[7]Lyon, D. A. Keeney, R. M., and Cullen, R. F.: The Electric Furnace in Metallurgical Work, Bulletin No. 77, U. S. Bureau of Mines (1914).

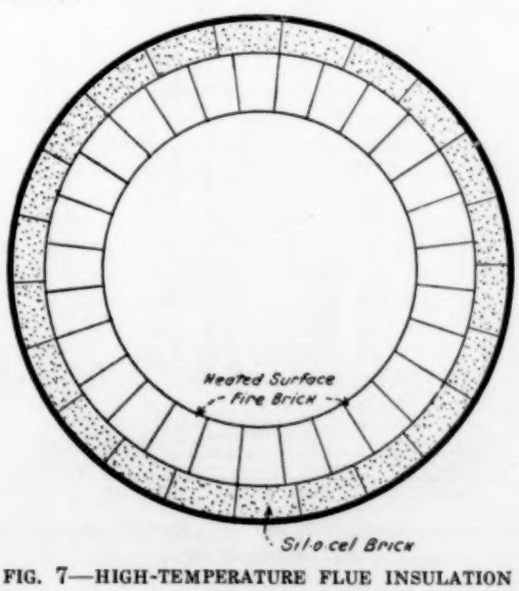


FIG. 7—HIGH-TEMPERATURE FLUE INSULATION

to approximately 30 tons capacity and for a product treated to even less heat, the capacity is increased as high as 40 tons, the heat being the same in all cases.

In electric-furnace practice, where extremely high temperatures are encountered and where effective insulation is necessary because of the relatively high cost of current, the question of insulation, or rather over-insulation, has been given more attention than in other lines. F. T. Snyder[1] has presented a detailed study of the nature and amount of insulation which is permissible under various conditions and presents many valuable data which might with advantage be applied to other industries.

DESCRIPTION OF CELITE

The requirements of an ideal insulator have been outlined, and in order to show the various methods of application which have been worked out, it will be well to select the insulator which apparently most nearly meets the requirements and use this as an example of methods of construction. In the applications shown, an attempt was made to cover as wide a field of use as possible, to show the scope of possible applications.

The mineral known as celite, on account of its extremely cellular nature, is a mineral of a highly siliceous composition and of very light weight, which occurs on the Pacific Coast[2] in an exceptionally pure state. It is composed of numerous hollow cells, and weighs in its natural rock-form, air dried, from 25 to 30 lb. per cubic foot. When this material is ground properly, it weighs but 8 lb. to the cubic foot, and has a thermal insulating power of about 9 to 12 times the insulating power of ordinary firebrick. In other words, a 1-in. layer of this material is the equivalent in insulating value of from 9 to 12 in. of firebrick. Being almost pure silica, its melting point is high, 2930 deg. Fahr. (1610 deg. C.), as reported by the Bureau of Standards, and it can be subjected to high temperatures without fear of alteration.

It has been found advisable, however, not to use celite as a refractory at extremely high temperatures without some direct protection. This is readily accomplished by using it as a backing material for more refractory and highly conducting bodies. Owing to its remarkable non-conducting properties, the accumulation of heat on its face is so great due to the fact that the surface is not cooled by conduction, that a "flash" of flame or gases might easily exceed the melting point of silica and cause failure. If it is protected, however, only modified and uniform temperatures are encountered, which are maintained without damage.

It is possible further to prepare bricks and blocks of various sizes and shapes by sawing the natural material by means of gang saws. Standard 9-in. straight silocel brick made from natural celite weigh from 1½ to 2 lb. each and are equivalent in insulating value to many times their thickness of ordinary firebrick. In crushing strength these bricks withstand over 400 lb. per square inch, and are sufficiently strong to stand transportation and handling.

The high insulating value of silocel bricks can be shown by applying the heat of a blast lamp or torch upon their surface for hours, the unheated surface remaining cool enough to allow handling with ease. Powdered silocel can be tested in the same way by packing it into a shallow paper box of convenient size.

The cost of these insulating bricks is but little more than that of firebrick and that of the powder about one-third as much, so that the first cost of insulation is comparatively low. In fact, instances are on record where the entire cost of insulation has been saved in fuel in the first few months of operation, leaving the gain in equipment efficiency and increased output or capacity as a clear profit.

[1] F. T. Snyder: The Flow of Heat Through Furnace Walls, Transactions of the American Electro-Chemical Society, vol. xviii, p. 235 (1910).

[2] Deposits worked by the Kieselguhr Company of America, New York, Chicago, San Francisco and Los Angeles. See the article by the author in *Metallurgical and Chemical Engineering*, vol. xii, p. 109; Feb., 1914.

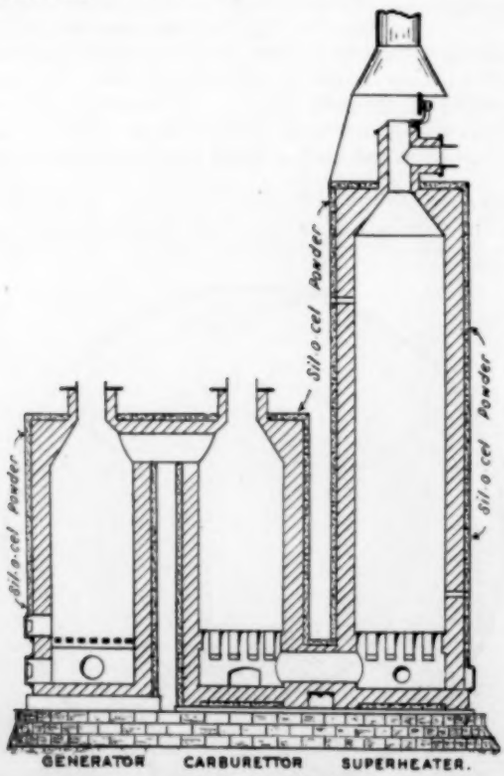


FIG. 8—GAS GENERATOR SET INSULATED BY MEANS OF POWDERED SILOCEL

TYPICAL WALL INSULATION

Because of the variation in forms in which silocel products are supplied—that of brick, blocks of various shapes, in powdered form and as a plastic cement—this material is adaptable to almost any form of thermal insulation, as will be shown in the following typical examples:

GENERAL TYPES

In general, there are four forms of construction for high temperature insulation which can be adapted to almost any character of equipment.

Fig. 1 indicates the usual method of using silocel brick interlaid between a course of firebrick and red brick for the prevention of heat leakage through walls. This form of construction is largely used in boiler settings, bakers' ovens, reverberatory furnace walls and roofs, etc., and is generally applicable where a strong, solid non-conducting wall is desired. Metal bonds can, of course, be used if desired, and the insulating brick built up in a uniform course.

Fig. 2 indicates one of the methods of construction of an insulating wall in which an otherwise hollow space is filled with insulating powder. From 2 to 4 in. are usually sufficient. The powder is packed slightly to a density of approximately 12 lb. to the cubic foot, at which point it attains its maximum insulating value and is not subjected to settling or contraction due to either vibration or heat. Where this form of construction has been in severe service in high-temperature furnaces for a period of years, no contraction or settling has taken place.

Fig. 3 illustrates the insulation of a boiler setting by means of silocel powder in hollow wall (left hand of diagram) and by means of silocel insulating brick (right hand and top of diagram).

Figs. 4 and 5 indicate the methods of insulating brick walls which are already in place. This form of insulation can be applied to old construction as well as new. In this method expanded metal lath is erected on an angle iron at the required distance from the outer wall and coated on the outside with one or more coats of Portland cement plaster, to which a small amount of silocel powder, approximately 20 per cent by volume, has been added to give greater plasticity and ease of working and to increase the heat-resisting properties of the cement. Silocel powder is packed to a density of 12 to 15 lb. per cubic foot between the brick wall and the expanded metal lath. This form of construction is relatively inexpensive and allows of as much insulation as is required, and furthermore gives an absolutely permanent surface of excellent appearance, which can be applied to almost any character of equipment.

INSULATION OF HOT-BLAST STOVES

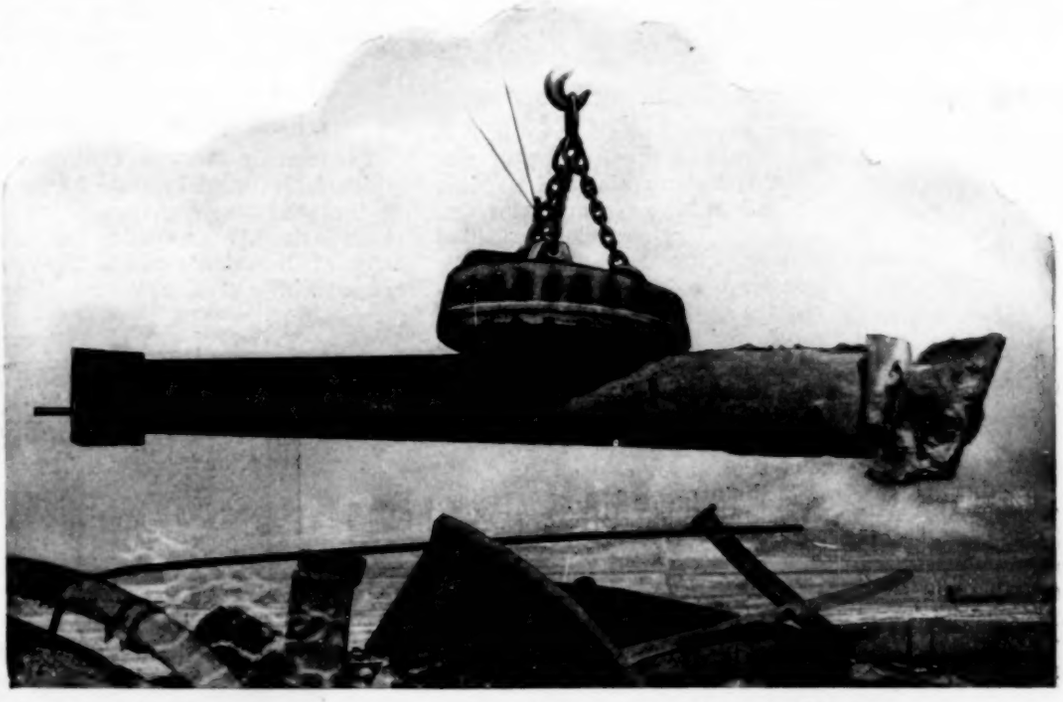
Hot-blast stoves (Fig. 6) and mains (Fig. 7) serving a blast furnace give an interesting application of high-temperature insulation, a number of these having been insulated in the Pittsburgh and Cleveland districts with excellent results. The detailed figures relating to the increased capacity and the thermal data are being compiled and will form the subject of another paper. At the present time, complete figures are not available.

Both insulating celite brick and celite powder have been used for this purpose with apparent equal beneficial effect.

The advantages derived from the effective insulation of stoves and high-temperature flues have been pointed out by Maccoun, Johnson, Mathesius, Wysor and others, it being estimated that preventable heat losses from surface radiation varies from 8 to as much as 20 per cent.

Hot-blast stoves provide an excellent example of the requirements of an insulation for high-temperature work, not so much because of the high temperature to which they are subjected, but more especially because of the size, expansion and contraction of lining and shell, settling, shrinkage, elasticity and corrosion of the insulator and protection of the shell itself, the constant change in temperature and other factors.

In a 20 x 100-ft. stove, for instance, the column of brick in the lining may expand through heat as much as 5 or 6 in. in height from cold to maximum heat. Therefore, the insulator may be considered at all times being subjected to a certain amount of stress and friction requiring an elastic medium which will take up these continued strains without injury or danger of being crushed which would result as is the case of granular slag in constant settling, the accumulation of fine material causing occasional bulging of the shell at the base of the stove on account of its inelastic nature. Two principal requirements of the insulator are, then, elasticity with still sufficient strength to give the proper support to the brickwork and shell, and, secondly, the proper amount of cohesion or high internal coefficient of friction and light weight as to overcome any tendency toward settling under the movement or vibration of the stove.



52-IN. MAGNET LIFTING SCRAP CAST-IRON COLUMN WEIGHING ABOUT 8 TONS

Lifting Magnets of Large Capacity

The lifting magnet has been adapted for the handling of materials in all branches of the iron and steel industry and has exerted a great influence in reducing the cost of handling many materials. Magnets designed for large lifting capacity in handling raw material have been placed on the market by the Cutler-Hammer Clutch Co., Milwaukee. These magnets are known as "High Duty" magnets and are designed especially for handling raw material. Recent improvements have increased the lifting capacity from 20 per cent for some sizes up to 60 per cent for others.

They are made in various sizes ranging from 18 in. to 62 in., the approximate lifting capacity of the 62-in. magnet being 4000 to 4500 lb., while in certain cases where a good surface is afforded the magnet will lift as high as 60,000 lb. Under adverse conditions the lifting capacity may drop to 2000 lb. For the 62-in. magnet a current of 72 amp. at 220 volts is required.

The frame or body of the magnet is made of a special grade of dynamo steel. The coils in the very small magnets may be made of wire, but for the larger magnets the highest space efficiency is obtained by constructing the coil of alternate turns of copper strap and fireproof insulating ribbon. The spool is of steel and forms a part of the inner pole. An absolutely tight

joint between spool and magnet case is obtained, thus eliminating all dead air space and facilitating rapid radiation of heat. It is necessary to design the magnet so that the heat of the coil may be quickly and easily conducted to the outside of the body from which it is radiated. This is effected by removing dead air spaces.

Another feature of the magnets is their waterproof construction. This is accomplished by first fully assembling the magnet, and when all parts are in their proper position, it is subjected to a baking process which is continued until all traces of moisture in the interior of the magnet have been eliminated. Air is next exhausted from the casing and a hot liquid filler is forced in under pressure, which presently hardens. One of these magnets was successfully operated 70 ft. below surface in the Mississippi River.

The magnets handle heavy castings, scrap, pig, and a great variety of other material.

Simplifying the Ore-Crushing Process

Simplification in all departments of ore treatment is noticeable in modern design and construction of machinery and plant. In the field of crushing and grinding we observe the simple ball and pebble mills for fine grinding, while for intermediate or secondary crushing there is a tendency away from the use of the small jaw or gyratory crushers or large rolls. These latter machines lose capacity rapidly as the required fineness of their product is increased, and they are readily choked by overfeeding. An efficient substitute for these machines is found in the Symons disk crusher, made by Chalmers & Williams, Chicago Heights, Ill., who present the following data from an investigation by Robert S. Lewis of Salt Lake City, Utah.

In the field of intermediate or secondary crushing, meaning the further reduction of the product of coarse breakers—a product from 3 in. to 6 in. in size—the common practice has been to use smaller jar or gyratory crushers or large rolls. The tendency toward simplification is well illustrated in the late type of Symons disk crusher, which is particularly suited to this work of secondary crushing. Its freedom from heavy vibration, few and long-lived wearing parts, true crushing action, uniformity of product which contains very little oversize, and low first cost are bringing this machine into wide favor with mill men.

The crushing members are two dish-shaped disks of manganese steel placed with their hollow sides facing each other. These disks rotate in the same direction and at the same speed. They are supported at an angle to each other, so that there is a wider opening between the edges of the discs at one part of their circumference than at the opposite part. The material to be crushed is fed into the space between the disks through a central feed opening in one of them. Centrifugal force then throws the material into the space where the disks are widest apart, and on being carried around it is crushed as the disks come closer together. The action is a true crushing one and not a combination of crushing and grinding. Particles sufficiently fine are thrown out between the disks, while the larger particles are again subjected to crushing. The disks can be adjusted to compensate for the wear, and thus a uniform product can be maintained.

Sizes and capacities of these disk crushers are given in Table I.

Cost for repairs and renewals ranges from 0.2 cent to 0.6 cent per ton of ore crushed. In one case a set of disks crushed 130,000 tons of hard rock and gravel from 6 in. to 1 in. In another case, crushing ore, the record for two different sets of disks was 155,385 and 200,812 tons, at costs of $0.0014 and $0.00109 per ton, respectively, for disks. These records were made on 48-in. crushers, handling 45 tons per hour, crushing from 3 in. to ⅞ in. and requiring 35 hp. each. The total cost of operation for a period of eleven months was $0.01598 per ton crushed, including an extraordinary cost of $0.00228 per ton due to steel getting into the crusher, leaving a normal cost of $0.0137 per ton crushed.

TABLE I

Size of crusher..	18 in.		24 in.		36 in.		48 in.	
Size feed opening	1.5 in.		2.5 in.		3.5 in.		6.4 in.	
Minimum exit opening and tons per hour for best results	⅜"	5-8	½"	12-15	¾"	25-30	1"	45-60
Minimum capacity, tons per hour, when crushing to size given	½"	8-10	¾"	18-20	1"	30-45	1½"	60-80
	¾"	10-12	1"	20-25	1½"	45-60	2"	80-100
	1"	12-15	1½"	25-30	2"	50-60	2½"	100-120
Horsepower required	12-18		18-25		30-40		50-65	

Chain Vernier Analytical Balance

The recently developed Christian Becker chain balance which is featured by its extreme simplicity, is shown in the accompanying illustration. The balance has a chain fastened at one end to the right side of the moving beam of the balance and at the other end to a slide arranged with a vernier. As this slide is raised or lowered

CHAIN BALANCE

the pull on the beam is increased or decreased, thus creating a series of different weights read from a scale on which the slide moves.

The chain is of gold and the slide is operated by a thumb screw from the outside of the balance so that weighing can be done with the window closed. The saving in time effected by simply reading the vernier and the dong away of small weight transferring are also important factors. Troublesome riders and small weights up to 50 mg. are eliminated.

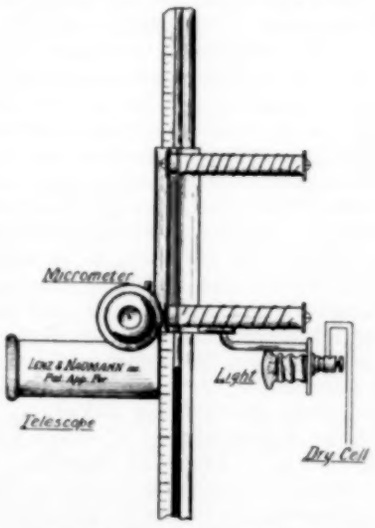
The balances are sensitive to 1/20 mg. They are sold by Eimer & Amend, New York City.

The Manufacture and Uses of Alloy Steels is the title of Bulletin No. 100, United States Bureau of Mines, a timely publication on the properties and methods of manufacture of materials that are coming into wide metallurgical use. The chemical composition and physical properties of different alloy steels are given, both for those that find extensive use and others that are less well known at the present time.

Micrometer Reading Device

A reading device for thermometers and burettes which comprises a mechanically adjusted micrometer reading feature in addition to the magnifying lenses is shown in the accompanying illustration. The optical equipment consists of a combination of lenses, and in addition there is an adjustable lens with a very fine hair line through its center and connected to a graduated micrometer screw in such a way that it may be moved across a certain space in either direction. This makes it possible to subdivide the space between any two graduations of a thermometer, burette, etc., in from two to twenty parts. A small electric-light bulb which comes in back of the thermometer, burette, etc., facilitates the readings. The device is very light, being made of aluminum throughout.

In order to compensate for the error due to parallax a line is made on the small electric-light bulb so that three lines are available, the one on the lens, the line to be read on the thermometer, and the line on the bulb.



MICROMETER READING DEVICE

If the lens is adjusted until these three coincide each time no errors will occur.

To give an example of using this device, say the mercury of a thermometer graduated in degrees registers somewhere between 97 and 98 deg. C. The scale on the head of the micrometer screw is turned to zero, the movable lens with its dividing line set parallel to the graduation mark at 98. By turning the head of the micrometer screw, the dividing line will move from 98 deg. to the next graduation at 97 deg. As soon as it is parallel with 97 deg., we stop turning the scale, which may now read 12. Now we turn back the micrometer screw till the dividing line is parallel with the mercury thread, which is somewhere between 97 and 98 deg. We read off once more the scale of the micrometer screw, which, say, is 8. As the total distance of the mercury from 97 was 12 minus 8, which equals 4, then the exact point of the mercury in the thormometer was 97 4/12 or 97 1/3, not 97½, as it would have appeared with an ordinary reading device.

This device is known as the Lenzman Micrometer Reading Device and is sold by the firm of Lenz & Naumann, Inc., Pullman Building, New York.

The **Standard Calorimeter Company**, East Moline, Ill., has issued catalog B, describing the Parr coal calorimeter, gas calorimeters, sulphur photometer and total carbon apparatus.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Aqueous Bath Apparatus

(*Continued*)

568,258, Sept. 22, 1896, Victor Joseph Kuess, of Bordeaux, France.

Relates to a process of and apparatus for distilling fatty substances, and consists of a steam-jacketed retort, the exhaust steam from which passes through the condenser to aid in drawing out the vapors from the retort. The retort is further internally heated, and the contents thereof treated with steam from a manifold having a plurality of perforations, steam issuing therefrom passing through the material to be distilled. The retort is also supplied with an electric resistance heating coil for internal heating, and also with a pair of parallel horizontal electrodes, preferably of aluminum, by which the ascending steam and fatty vapors may be subjected to electrolysis.

573,355, Dec. 15, 1896, Marshall Pridham, of Philadelphia, Pa., assignor, by direct and mesne assignments, to the Electric Rectifying and Refining Company of Camden, N. J.

Relates to apparatus for purifying and decolorizing saccharine liquids, water, etc., from nitrogenous, organic, or mechanical impurities, and consists of a preferably long rectangular tank containing electrodes; the tank is filled with the liquid to be purified by an injector which also thoroughly mixes the liquid with ozone. The liquid is electrolyzed by passing a current from an aluminum or zinc anode supporting a quantity of active material, such as aluminum hydrate, or zinc oxide, the active material passing to a negative electrode. Before electrolysis, the tank is exhausted of air or other gases, which removes entrained gases from the liquor. During electrolysis, a stream of ozone is injected, the ozone oxidizing the impurities, which settle to the bottom and may be drawn off. The aluminum hydrate also collects impurities under the action of the current, and settles either on the cathode, or at the bottom of the tank. If the liquor is not sufficiently purified, it is then sprayed into a cylinder containing air under pressure, the sprayed liquid being treated with ozone while entering the cylinder, and the liquid being treated with ozone by a perforated manifold, one section of which rotates in the cylinder. The liquid may then be subjected to a secondary treatment, by first exhausting the air, then admitting ozone through the manifold.

574,038, Dec. 29, 1896, Richard J. Marks, of Hartford, Conn., assignor of one-half to William P. O'Toole, of same place.

Relates to a work-holder for articles to be electroplated, the holder consisting of a pair of hinged wood frames, the top and bottom of which are covered with wire netting. The bottom covering is divided into four insulated sections, separated by wood cleats, the sections are connected to the negative wire from alternate ends, so as to provide a more uniform distribution of current to the articles to be plated.

591,730, Oct. 12, 1897, Willy Bein, of Berlin, Germany.

Relates to a process of and apparatus for electrolyzing solutions. The apparatus consists of a cell divided by impervious vertical partitions into two, three, or four compartments, the partitions not completely

separating the compartments from each other, but leaving a continuous liquid connection between them. With the four compartment cell, the two outer partitions extend downward from the top about three-fourths the distance of the cell, the intermediate partition extending upward a like distance. Above the intermediate partition, a depending adjustable gate is provided to vary the area of the liquid connection or passage. The outer compartments are covered, and in the electrolyte are placed horizontal electrodes near the surface thereof. Gaseous products pass off through vents, while products heavier than the electrolyte, such as sodium hydroxide, sink, and may be drawn off periodically. The electrolyte forms a fluid diaphragm between the anode and cathode, and is placed in the inner compartments; it is replenished as often as necessary. A modification consists in supplying a slow stream of electrolyte to the anode compartment, and allowing slow streams of anolyte and catholyte to siphon off from the electrodes. The patent should be consulted for details.

605,835, June 21, 1898, Emile Andreoli and Gabriel Andreoli, of London, England.

Relates to the electrolytic production of amalgams, and describes a horizontal cell consisting of two outer anode compartments, and a central cathode compartment, separated by porous partitions, the floor of which is covered with mercury. In the outer compartments are solutions of salt, and a layer of water covers the mercury to the height of the salt solutions. When the mercury is sufficiently rich in sodium, it may be withdrawn, and used for amalgamated copper plates to be used in the recovery of gold, etc., by amalgamation in gold mining. The central compartment may contain a mass of inert material, such as a block of cement, to provide a narrow channel for mercury next to the porous diaphragm, and thereby economize in mercury.

621,907, March 28, 1899, Herbert H. Dow, of Midland, Mich.

Relates to an electrolytic cell in which a layer of solution similar to the electrolyte surrounds a closed cell, thereby preventing any leaks of gas, etc.; or the layer of liquid may only cover the top of the cell. The liquid above the cell also filters into the pores of the carbon electrodes which project through the top of the cell, and into the cell, thereby preventing creeping of salts, etc., through the carbons.

625,489, May 23, 1899, William Y. Buck, of Bristol, Conn.

Relates to a work holder for electroplating, and consists of a metallic bar having alternate equally and oppositely inclined holes for the reception of articles to be plated, such as unmounted table knives. The tangs are inserted in the holes, the blades projecting upward and outward at an angle of approximately 50 deg. apart. The bar is suitably suspended in the electrolyte. During deposition, the more concentrated layers of electrolyte are at the bottom of the tank, and ordinarily this would result in a heavy deposit upon the tangs of the knives; in practice, however, the tangs exert a shadowing action upon each other, resulting in a relatively weak deposit. Since the blades are widely separated, they do not exert a shadowing action upon each other, and receive a normal deposit.

637,313, Nov. 21, 1899, S. Wicks, of Cleveland, Ohio.

Relates to an apparatus for sharpening files, and consists of a lead-lined tank containing a suitable acid electrolyte. The tank contains a plurality of vertical pockets in its walls, having open tops and bottoms, and perforated shields therefore, to prevent accidental contact of the cathodes within said pockets, and the files, which are connected as anodes. The files are separately electrically connected to the anode wire by a switch, and are adjustably suspended vertically in the electrolyte.

645,785, March 20, 1900, W. Y. Buck, Bristol, Conn.

Relates to holders for electroplating spoons, etc., and consists of a pair of wires, twisted together at intervals, and leaving a plurality of rectangular loops between the twisted sections. The twisted wires are now bent in the shape of a rectangle, having a length several times its width, and provided with a handle or bail by which it is to be suspended in the electrolyte and electrically connected to the negative wire. The spoons to be plated are inserted in the loops, with the inner sides of the bowls facing toward the center of the frame. With this arrangement, the bowls receive a heavy plating on the outside, and a thinner deposit on the inside.

645,786, March 20, 1900, W Y. Buck, Bristol, Conn.

Relates to the art of electroplating, and covers the process of "shadowing" surfaces of articles to be electroplated, thereby effecting a thin deposit on the shadowed surfaces, while producing a thick deposit upon the exposed portions. This patent is for the process used with the holder described in patent No. 645,785.

647,960, April 24, 1900, George Weltden Gesner, of New York, N. Y., assignor to Harleston Corbelt Gesner.

Relates to an electrolytic cell comprising a vessel and an anode and cathode of an alloy of iron and hydrogen, the hydrogen being present in such quantity as to prevent all oxidizing action, including that arising from chlorin. The patent refers to several other patents and applications covering the alloy and processes for its manufacture. Several methods are briefly referred to in this patent, one of which is to force hydrogen through the molten metal, and then to remove the alloy that has been formed from the unalloyed iron either by puddling and squeezing, or by grinding and sifting. The alloy may contain hydrogen in the proportion of eleven one-hundredths of 1 per cent and upward. A cell comprising an anode and a cathode of this alloy is adapted for the decomposition of saline solutions, and salts which contain an excess of chlorine. The alloy is said to be capable of resisting oxidation and corrosion from heat or chemical action arising from either atmospheric or more powerful influences.

649,614, May 15, 1900, Antoine Edouard Peyrusson, of Limoges, France.

Relates to an electrolytic cell for the decomposition of electrolytes, and consists of concentric electrodes, the inner one contained in a porous cup which is provided with a faucet at the bottom, the outer one surrounding the porous cup and contained in an outer cell also provided with a faucet at the bottom. Both electrodes are in the form of a helix, and both fit tightly against the walls of the porous cup, and the cell. Fluids to be electrolysed are admitted from faucets above, and travel in helical paths between the convolutions of the helical electrodes, the spaces between the convolutions constituting electrolyte compartments of considerable length. The velocity of flow of electrolyte may be controlled by the faucets at the bottoms of the inner and outer cells.

675,662, June 4, 1901, Samuel Lowe, of Waterbury, Conn.

Relates to a rack to support articles to be electroplated, and consists of a central vertical rod provided with a hook at the upper end, and a plurality of horizontal arms secured thereto, and extending each side thereof, and upon which the articles to be plated are placed. The upper horizontal arm has its ends bent upwards, while the lower arm has its ends bent downwards. In order to prevent the articles to be plated from slipping off the arms, end-pieces are provided having openings to receive the extremities of the arms, the upper and lower arms being sprung into place, their bent ends securely holding the end-pieces.

Industrial Notes

The Hooker Electrochemical Company's new plant at Niagara Falls is now in rapid course of erection by the Samuel Austin & Son Company. Probably 700 men are on the job now; at night, from 150 to 200. The entire work is being handled on a "cost-plus" basis. There is an engineering force of over sixteen men on the job, in charge of Mr. H. F. Jacobi. There is also a complete accounting force under an auditor, so that Mr. Hooker has on his desk each day at 9 a. m. cost reports covering each unit of work for the previous day.

C. W. Leavitt & Co., New York, have been appointed exclusive American selling agents for the Carlton Iron Company, English manufacturers of ferromanganese.

A dyestuff plant will be erected by Messrs. Marden, Orth & Hastings on a plot of land comprising ten acres in the meadows near Jersey City.

The Oliver Quartz Co. has taken over the business of T. C. Oliver, Charlotte, N. C. The quartz sold by this company has been used in packing a great many of the acid towers in eastern chemical plants.

The Pacific Coast Steel Company recently closed contracts for extensive enlargements to its plant to provide for the manufacture of structural steel on a large scale.

Bavarian Porcelain.—Messrs. Lenz & Naumann, Inc., New York, announce that they still have a good stock on hand of imported Bauscher Bavarian porcelain which is said to equal the Royal Berlin in acid, alkali and heat-resisting qualities.

The American Steam Pump Co., Battle Creek, Mich., has recently received an order for all the pumps to be installed in the new Curtis Bay distillery of the Republic Distilling Co., Baltimore. The order includes pumps for fusel oil, alcohol, gasoline, fresh and salt water, etc.

The Peruvian Potash and Chemical Co., Philadelphia, Pa., recently incorporated, intend to develop potash deposits in Peru. These deposits are stated to have lain for centuries untouched, with only a 3-ft. overburden of soft sand and gravel. The deposits lie on the southern boundary line of Peru, and are ten miles inland from the sea. Shipments to the United States are contemplated about April 1.

The National Ox-Hydric Company of Chicago, Ill., will install commercial oxygen plants for the sale of oxygen and hydrogen in Chicago, Pittsburgh, New York, Boston, Minneapolis, St. Louis, Kansas City, Detroit, Philadelphia and Los Angeles. Machinery and equipment for these plants is all completed and considerable of it has been shipped. The company will use its own generating system.

Japanese Electrochemical Works.—The British vice-consul at Osaka reports that a company has been formed in Osaka with a capital of 350,000 yen (about $178,500 at par) for the manufacture of caustic soda and bleach by the electrolysis of common salt. A factory is to be erected at Kyushiu where hydroelectric power is fairly cheap. The machinery is to be Japanese. A monthly output of 300,000 lb. of caustic soda is spoken of.

The Snyder Electric Furnace Co. of Chicago, Ill., announces sales of electric steel furnaces during 1915 from their English office at Bradford, Yorkshire, England, to the following companies: The Daimler Co., Ltd., England; Thwaites Bros., England; Summerson & Sons, Ltd., England; National Steel Foundry Co., Ltd., Scotland. In addition, four large furnaces for special smelting work have been contracted for as well as a small furnace for carrying on initial experimental smelting work.

Magnetic Separator Pulley.—A recent circular of the Cutler-Hammer Clutch Co. of Milwaukee describes its magnetic separator pulley for use in paper and pulp mills, cement plants, gypsum plants, starch mills, foundries, steel plants, mines, glass factories, rubber plants and other industries. This magnetic pulley is designed to remove stray pieces of iron and steel and is of particular value where crushing or pulverizing machinery is used, as by removing the nuts, bolts, pick-heads, etc., it prevents injury to the crusher blades. In other industries it is used to remove small particles that detract from the purity or quality of the product. An example of the latter is the sugar refinery.

Chalmers & Williams, Inc., of Chicago Heights, Ill., have received an order from the New Cornelia Copper Co. at Ajo, Ariz., for twelve 48-in. Symons fine reduction disc crushers, which will be used in their plant to reduce 4000 tons from 4 in. to ¼ in. Orders were also recently received for similar machines from the Chile Exploration Co., Hollinger Gold Mines Co., and C. S. Christensen, Kristiania, Norway.

The Apollo Electric Steel Co. has been organized for the manufacture of electric furnace steel at Apollo, Pa. A site of 8½ acres has been secured and the contract let for suitable steel buildings. They will be of the most modern construction and will be equipped with every facility for the manufacture of high-grade steel. Two Snyder electric furnaces will be used, each having a capacity of 100 tons per day. The management of the company will be in the hands of the following men: Robert Lock, president, formerly with Allegheny Steel Co. and American Sheet & Tin Plate Co.; J. Arthur White, metallurgical engineer, formerly with American Sheet & Tin Plate Co.; W. E. Troutman, manager of Vandegrift plant of United Engineering & Foundry Co.; J. E. Gallagher, of Apollo, and Carl H. Booth, vice-president of the Snyder Electric Furnace Co., Chicago.

Standard Tables for Petroleum Oils.—Samples of petroleum oil have been collected from different parts of the country by the United States Bureau of Standards, and the specific gravity determined over a wide range of temperature. From the data obtained tables have been prepared for determining the true specific gravity and volume of oil at the standard temperature, when these quantities are measured at other temperatures. Tables have also been prepared for showing the relation between specific gravity, Baumé degrees, and pounds per gallon. The new tables will be especially useful in determining the quantity of oil in large shipments. Circular No. 57 of the Bureau of Standards presents the results of these experiments, and it may be obtained by interested persons from that Bureau.

Measuring Barium Compounds in Rubber Goods.—By means of tests made on compounds of known composition, prepared at the U. S. Bureau of Standards, a method has been devised which permits the quantitative determination of barium carbonate in the presence of either lead sulphate or barium sulphate, the two sulphates most commonly used in rubber goods. This is of help in determining the total sulphur when both barium carbonate and sulphate are used as fillers. These tests are described in Technologic paper No. 64, which may be obtained from the Bureau of Standards.

The Riche Adiabatic Calorimeter, sold by Messrs. Lenz & Naumann, Inc., of New York, and described in this journal September, 1914, page 606, is reported to find increasing application where accurate results are required, on account of the many interesting features which make it an accurate measuring instrument.

Personal

Mr. **A. A. Cole** is the nominee for president of the Canadian Mining Institute. He has been known for his work as mining engineer for the Temiskaming & Northern Ontario railroad, with particular reference to the mining and metallurgy of the Cobalt and Porcupine districts.

Mr. **Herbert Coward,** who for three years has been in charge of the Spiro Turbine Department of the Buffalo Forge Co., and for four years previous engaged in sales and engineering work in connection with air conditioning apparatus, has been appointed steam engineer of the Carrier Engineering Corporation and is now located at its main offices, 39 Cortlandt Street, New York.

Mr. **A. M. Douglass,** formerly chemist in charge of the laboratories of the Merrimac Chemical and the New England Manufacturing companies of Boston, Mass., is now engaged in the installation of the plant and process of the American Bromine Company, located at Midland, Mich.

Mr. **W. B. Easton,** chief engineer for Chalmers & Williams, is visiting Arizona mills in the interest of his company.

Mr. **Justin H. Haynes** and Mr. **Edmund Leet,** who have been in the Cripple Creek district for the past five months, have completed their experimental work on flotation for the Vindicator Consolidated Gold Mining Co.

Mr. **Henry Howard,** vice-president of the Merrimac Chemical Co., delivered an address on the "Necessity for an American Dyestuffs Industry to Aid Export Trade in Textiles" at the National Foreign Trade Convention at New Orleans on Jan. 27-29.

Mr. **H. C. Humphrey,** chief chemist of the Eastern Branch of the Corn Products Refining Company in New York, died of pneumonia after a short illness.

Mr. **S. A. Ionides** has returned to Denver from a professional trip through the Northwest.

Mr. **B. T. Klein** has been appointed Pacific district manager of the Bristol Company. This is a new branch office recently established at 727 Rialto Building, San Francisco. Mr. Klein was Chicago manager for the company for six years and had charge of the company's exhibit at the Panama Exposition.

Mr. **E. W. Lawler** has again become associated with the Hardinge Conical Mill Co. and is located at the New York office. New quarters have been leased by the company at 120 Broadway in the Equitable Building, and it is expected that they will be at the new address by Feb. 20.

Mr. **H. S. Mulliken** is manager for the mines and smelter of the Penoles company at Mapimi, Durango, Mexico.

Dr. **Charles S. Palmer,** formerly professor of chemistry at the University of Colorado and later president of the Colorado School of Mines, has accepted a fellowship in the Mellon Institute for Industrial Research at Pittsburgh.

Mr. **W. J. Pentland,** general manager for the Amajac Mines Co., Hostotipaquillo, Jalisco, Mexico, is still at Guadalajara. In the last communication received from him, dated Dec. 28, 1915, he states that it has been impossible to carry on any work at the mines.

Mr. **Charles F. Rand** was elected president of the United Engineering Society at the annual meeting, Jan. 27, Mr. Gano Dunn declining re-election.

Dr. **L. D. Ricketts** has gone to South America to inspect recent developments in copper mining and metallurgy.

Mr. **Frank W. Steere,** vice-president of the Steere Engineering Co., Detroit, Mich., was awarded the "Beal Gold Medal" for the best paper read at the annual meeting in 1914 of the American Gas Institute. Mr. Steere's paper was entitled "An Electrical Process for Detarring Gas," and was published in our issue of December, 1914, Vol. XII, page 775.

Mr. **W. G. Swart** has transferred his office from Denver to Duluth, Minn., where he is engaged on investigations relating to the treatment of iron ores.

Mr. **S. W. Traylor,** president of the Traylor Engineering & Mfg. Co. of Allentown, Pa., has been chosen manager of the ordnance department of the Crucible Steel Company's plant at Harrison, N. J. The new position will not interfere with his old one.

Mr. **Thomas F. Williams,** formerly chief mechanical engineer of the Aeromarine Plane & Motor Co., of Avondale, N. J., and a specialist in carburization of fuels, has accepted the position of chief consultant of the Board of Engineering Research, Mechanical Applications, of the Powdered Coal Engineering & Equipment Company of Chicago and will direct the research work of that company in the carburization of comminuted fuel and the application of automatic mechanical regulation and controls thereof.

Mr. **Louis A. Wright** has severed his connection with the General Development Co. and has joined the staff of the American Metal Co. He has sailed for Chile, which will be the scene of his activities for the next two years.

Book Reviews

Inventions and Patents. By **Philip E. Edelman.** 288 pages. Price $1.50, New York: D. Van Nostrand Company.

This book is intended to be a practical guide to the inexperienced inventor who has not had the opportunity to know of the numerous obstructions in the way of applying for and securing U. S. patents. The author devotes a few pages to the development of the patent system and then in considerable detail describes the methods to pursue in choosing an attorney, making applications, patent office practice, methods of protection, sale of patents and so on. An appendix gives a digest of important patent decisions and the various forms used in patent procedure and closes with a short bibliography of the general subject.

The author has carried out his intentions in a successful manner, his style is easy, the language simple and non-technical. The book is well printed in good type and deserves a place on the shelves of every manufacturer.

Autogenous Welding and Cutting. By **Theodore Kautny,** translated from the German by the author and **James F. Whiteford.** 157 pages. Price $1.00, New York: McGraw-Hill Book Company, Inc.

This handbook of the art of welding is a recent résumé of the latest practice in Germany and is intended to go directly into the hands of the operator in the works. It describes the methods of obtaining the various gases used, their purification, containers, pressure, regulation, piping and fittings. The author then devotes some pages to a description of blowpipes of various kinds and exactly describes their methods of use. Various kinds of metal are then taken up and the user is told just how to make vessels and articles of any shape. Copious illustrations assist the unusually clear description of operations and it is hard to see how a workman with average intelligence can fail to improve his technique by an occasional reference to this book.

The author is to be congratulated on laying before the public in so practical a way what has been too often considered in the light of trade secrets. No one who contemplates adding a welding department to his equipment should proceed before he has well studied this book.